COLLIERY VENTILATION

ALAN BAGOT

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COLLIERY VENTILATION

By the same Author.

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THEIR CAUSES AND PREVENTION.

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'A mining engineer can scarcely perform a more praiseworthy or useful task, than Mr. Bagot has accomplished in this little work. Accidents in mines are so common, so fatal, yet so capable of prevention, that those who can exert any influence upon people concerned ought to make it a duty to agitate the question in season and out of season. We therefore welcome the present work as a handy compendium, from the pen of a practical man, of all circumstances tending to produce accidents in coal mines, of the precautions that should be taken to prevent them, and of the remedies that should be adopted after them. . . . The book should be studied by everyone who has charge of a coal mine '—Iron.

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THE PRINCIPLES

OF

COLLIERY VENTILATION

BY

ALAN BAGOT, C.E.

MEMB. INST. MECH. ENGINEERS

SECOND EDITION, GREATLY ENLARGED

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LONDON
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PREFACE.

The former edition of this book being out of print, I have taken this opportunity of considerably enlarging the subject-matter in preparing this edition. The points on which the management of collieries in this country is assailable are, in the writer's opinion, insufficient precaution against blowers of gas and absolute want of any appliances ready at hand in case of explosion. I have given what particulars I think can be of service to the officials of a colliery respecting the course to be pursued after accident, but I wish to draw the attention of all those engaged in mining fiery pits to the risk attending the use of Davy and Clanny lamps.

Her Majesty's Commissioners appointed to inquire into mining accidents in their 1881 Parliamentary Report draw attention to this risk, but I think that this report will be but little heeded, judging from experience, inasmuch as on April 25, 1879, I read a paper before the Institution of Mechanical Engineers on the subject, with experiments proving the defects in Davy's lamp, and many other modified forms of it in use in mines; and in a work of mine published in 1878 I state the fact that the Davy lamp will explode in an explosive mixture travelling at a velocity of eight feet per second. Sir J. Eardley Wilmot, M.P., in reply to a question he asked in the House of Commons respecting the suggestion contained in that work, was told that the Government Inspectors of Mines had been provided with the book, and the Home Secretary had had hardly time to consider these suggestions. Between that period and the report of the Commissioners some hundred lives have been lost in coal mines through explosion. If the Government will not be convinced of the folly of sanctioning the use of Davy, Clanny, and all non-extinguishing lamps in mines, the only chance to avoid these disastrous explosions is to appeal to the common sense of the mining engineers. Several colliery companies whom I have advised in the matter of lamps have had trouble with the men with the compulsory substitution of extinguishing lamps, but in the end all has ended smoothly. I conclude that an Act will ultimately be passed to make the use of non-extinguishing lamps illegal, but why have the Government persistently ignored the fact, to which their attention was officially drawn in 1878 by Sir J. Eardley Wilmot, up to the present time? On the other hand, private individuals have expended time and money in endeavouring to make coal mining safer to those engaged in it. A considerable sum of money was expended in 1880 by His Grace the Duke of Northumberland on this head in experiments entrusted to my charge, and a large proportion of the formulæ in this book were deduced from such experiments.

With respect to the formation of life brigades in mining centres, the Government has done absolutely nothing. After an explosion there is generally at least two hours to wait before anything can be done, and it would be found far cheaper and better to have one set of apparatus for the district kept in a central position ready for use, than to trust to the individual resources of each colliery. I maintain, in spite of what has been said to the contrary, that if the aërophores, such as, by preference, Fleuss's or Denayrouze's, were available, with trained men, in a mining district, more lives could be saved at less risk to those whose duty compels them to make the attempt after explosion than at present, when exploring parties are driven back time after time by choke-damp and after-damp, and valuable moments are lost and their lives needlessly endangered.

The vacuum fan is gradually replacing the old furnace system of ventilation, and thus a source of explosion is prevented; but the explosions that have to be guarded against are those that arise from scaffolds over sump holes, air sheets left down that ought to be kept up, or *vice versâ*, and falls of roof. Individual forethought and precaution are the only preventitives of such sources of accident. Frequent inspection, on the part of competent men, of every district of the mine, and written instructions clearly expressed to those who have to attend to air crossings, doors and sheets, are imperative if accident is to be avoided.

The attention of the reader is drawn to the excessive pressures that are obtained by inserting tubes into the face of the coal, and to the importance of even air splits if possible.

The use of fire-damp indicators to detect the presence of minute quantities of gas I am opposed to. Competent firemen should be told off for this purpose, and the gas book entered regularly and not worked up afterwards during the fireman's leisure. The instant gas is found coming off in any quantity it should be constantly watched; especially is the necessity of this to be recognised when the roof does not pack up readily by falling to the solid rock. It is in these cavities between the rock and the temporary roof that gas collects, and its presence is never discovered till a heavy fall takes place, when the workings are filled with gas. The Abercarne colliery explosion illustrates this very clearly. It must not be supposed that the ventilation of a colliery is to be learnt in a book; all this treatise is intended to do is to give the theory of the usual system of ventilation, an abstract of the legislation on the subject, and such tables and formulæ as are constantly required by a mining engineer or underground manager. The practice of ventilation is best acquired by going underground regularly in the several capacities of surveyor, fireman, deputy, underlooker, and finally certificated manager, and thoroughly mastering the details and difficulties presented by each department.

ALAN BAGOT.

MARCH 1882.

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THE
VENTILATION OF MINES.

CHAPTER I.

GENERAL PRINCIPLES OF VENTILATION IN COLLIERIES.

There are two general methods of ventilation in collieries; one by means of a furnace at the foot of the upcast shaft, the other by means of a vacuum fan at the top of the upcast shaft, which latter system (by reason of the depression caused in the vacuum chamber, brought about by the rapid rotation of the fan) sucks the air out of the pit, or more accurately speaking, causes the air in the upcast shaft to ascend to fill the depression in the vacuum chamber, and so permits of a current of air flowing through the pit.

The action of the furnace system in producing a current of air is as follows:—A mine is simply a U-shaped tube, and it will be at once seen that the two upright columns of air in the arms of the tube must balance each other, or if one is lighter than the other the heavier column will descend and push the other up until equilibrium is produced, when the motion of the respective air columns will cease until further alteration takes place in the respective weights of the two columns, and it will be seen that if the difference in weight of the two columns is constant, the motion and the velocity of the whole of the air in the tube will also continue constant, and in one and the same direction.

We propose treating of the furnace system first. This system is only an artificial development of the natural laws of ventilation.

If air is heated under a constant pressure it expands $\frac{1}{459}$ of its volume at the temperature of zero of Fahrenheit's thermometer; and it can easily be understood that if the mean temperature of the upcast shaft is artificially increased, the heated air becomes lighter, bulk for bulk, than that in the downcast shaft; the consequence being that the heavier air column in the downcast shaft overcomes the lighter air column in the upcast, forcing the latter up the shaft before it; and if we were to prolong the upcast shaft to an indefinite length, the top of the heated column of air would stand possibly 60 feet higher than the level of the top of the cold air column. This difference is called the 'Head of Air.' The cool air, taking the place of the heated air, passes over the furnace in its turn, and so a perpetual current is kept up by the cool air going down one shaft and forcing the heated air up the other before it. It will be observed that the deeper the upcast shaft the more rapid will the draught be, and that the secret of a brisk circulation of air in the upcast shaft lies in the whole of the heat from the furnace being applied directly to increase by its action the temperature of the air column in the upcast shaft, and so set it in motion. The dumb drift should therefore be as short as possible, as far as is consistent with safety. The best method of laying out the furnace is as follows:—

Near the bottom of the upcast shaft the main return airway is split into two ways; the furnace is placed in a recess, cut out in the face of the smaller split of the main return airway, the main return itself enters the upcast shaft a little below the smaller drift; by means of a damper placed in front of the furnace, the amount of air necessary to support the combustion of the furnace is regulated. The result is that no more air than is actually necessary is consumed, and the remainder has access to the upcast, by way of the main drift beside the smaller one; by arranging the depth of the furnace below the entrance of the main return air-way and furnace drift, the supply of air and the temperature of the shaft may be so kept uniform that men may travel in the shaft. It must be borne in mind, however, that this system is fraught with danger.

In the first place, if the cage becomes fast in the shaft, the men would be suffocated, and the ventilation of the mine very seriously impaired, and any blower of gas that may be present in the pit would cease to become diluted, and would ultimately charge the partially choked upcast shaft like a gun-barrel, until it became ignited at the furnace. If, however, it is absolutely necessary to use the upcast shaft as a drawing shaft, then air doors should be fitted to the shaft to close at the furnace drift, and in the event of an accident they should be promptly closed and the fires damped out. Care must be taken in erecting these doors, to allow for the expansion of the iron guide blocks, or when the furnace is burning they will not close just at the critical moment. On the whole, travelling in the upcast shaft is an economical practice that should be avoided. It will naturally occur to the reader to ask, why the two air columns, if of the same temperature, but of different diameters, or rather sectional areas, balance each other. It must be evident, that the weight of a column of air, like that of a column of water, is directly as its height, and that whatever extra pressure exists on the area of the smaller column of air, by reason of the greater dimensions of the larger column, is really exerted, not on the sectional area of the smaller column, but on that part of the mine where the area of the larger column becomes diminished to the area of the smaller one, such as the sides of the air-ways.

In this way equilibrium is preserved. The elements of success, therefore, in furnace ventilation, are as follows:—

The upcast shaft should be free from obstruction, the deeper of the two, and every pound of coal consumed in the furnace should do its share of work in setting the hot air column in motion; no more air should be admitted to the furnace than is absolutely necessary to support combustion, and the temperatures in the two shafts should differ considerably, yet uniformly. Changes of temperature, or careless firing, will check the air current, and this is to be avoided. We must consider, therefore, some of the laws which govern the mechanical properties and effect of heat, in order to proportion our furnace to our requirements.

Coal is composed of carbon and hydrogen, and some other substances that are not necessary for our consideration. 2¼ lbs. of coal require 29 cubic feet of air to insure complete combustion; this is on the supposition that the whole of the oxygen contained in the air is converted by combination with the carbon into carbonic acid gas; but in practice it will be found that the best furnaces only burn half the oxygen of the air passing through them, so double the amount of air must be provided. The process of burning is this: the greater part of the carbon is transformed into carbonic acid gas, the remainder into carbonic oxide: the hydrogen combines with the remaining oxygen to form steam. A remarkable feature of the case is that at equal temperatures the volume of the gases resulting from combustion is nearly equal to that of the air passing through the furnace. It is, therefore, of no importance in calculating the effective force of a furnace, to ascertain whether all the air has been consumed in passing over the furnace or not. The same volume of air will be obtained whether the air has passed the furnace or not; that is to say, the quantity of air admitted to the furnace will not be appreciably diminished in bulk by passing through the furnace. From these facts the following formula has been obtained.

A volume of air V arriving at a temperature t at the furnace.

In the upcast shaft, when the temperature is represented by t^1, then $V = \frac{v^1 + a\,t^1}{1 + a\,t}$

This shows that it increases in volume, in a proportion due to its temperature, and to no other agency.

The next question is, what is the difference of height due to the difference of temperature in the two columns? In mining engineering the difference is called the *head of air*, or *motive column*, since it is the exact mathematical representation of the force, exerted to produce the motion of the two air currents. This case is found as follows :—

Let H = Head in feet.
D = Depth of downcast shaft in feet.
t = Temperature of downcast shaft.
T = Temperature of upcast shaft.

$$H = D \frac{T - t}{T + 459}$$

Or representing the difference of the two temperatures by x,

$$H = \frac{-Dx}{T + 459}$$

Or a simpler method of obtaining the same result is found as follows:—

Let D represent the length of the downcast column of air.
U = Length of upcast ditto.
T = Number of degrees in excess of 32° Fahr. in D.
t = Ditto in upcast.

$$U = D \left(\frac{480 + t}{480 + T}\right)$$

Then $U - D$ will equal the *head* H, and $8{\cdot}025 \sqrt{H}$ will give the velocity of the air current in feet per second in the upcast shaft.

This result H will represent the power we have from our furnace to impart motion to the air column, and can be converted to inches of water-gauge, as follows:—

The head of air H must be reduced to pressure in lbs. per square inch. The weight of air in the upcast shaft will vary, being affected by the temperature.

The weight W is thus calculated :—

$$W = \frac{1 \cdot 32529 \times 30}{459 + t}$$

$$W = \frac{39 \cdot 7587}{459 + t}$$

When t the temperature is 62° Fahrenheit the result is the weight of one cubic foot of air in decimals of 1 lb. avoirdupois. Then the head in feet must be multiplied by the weight of one cubic foot W, as obtained above.

$W \times H$ = pressure in lbs. per square foot of air of motive column.

The weight of one cubic foot of air at 75° Fahr. = ·0744544 lb. per square foot.

Referring to the formula under head of water-gauge, we find that W the water-gauge reading is thus calculated :—

$$W = \frac{\frac{1 \cdot 32529 \times 30}{459 + t} \times M}{5 \cdot 196}$$

M the motive column of air in feet is equal to the velocity in feet per second, squared and divided by 64·4.

A millimetre being 0·039370 of an inch, this may easily be converted to the French reading.

The calorific capacity of air was determined by M. Regnault, and is represented by 0·237. So that to raise 1 kilogramme of air 1° Centigrade, 0·237 kilo. of heat will be required.

The calorific power of coal varies with its composition, but to no very great extent; as a general rule it may be estimated that 1 lb. of coal will circulate 13,000 cubic feet of air. M. Devillez states that 1 kilogramme of small or slack coal will produce 7,500 units of heat; a kilogramme being 2 lbs. $3\frac{1}{4}$ oz.

It is now necessary to know what velocity is acquired for a given known difference in temperature between two shafts, or for a given head. This will be found as follows.

Let V represent the velocity required to be found in feet per second.

Let M be the height of the motive column in feet.

$$V = 8{\cdot}025 \sqrt{M}$$

The friction increases as the square of the velocity of the air current, and therefore the *velocity* of the air current will affect the *quantity* of coal consumed in the furnaces.

The causes of the loss of power with furnace ventilation are very numerous. M. Devillez proposes to allow double the necessary quantity of coal consumed as a compensation for the loss caused by the cooling action of the sides of the shaft, and the extra weight of the column of air in the upcast, due to the presence of carbonic acid gas and steam. From the foregoing, the principles of success in the construction of furnaces may be deduced, bearing in mind that from 13° to 20° F. variation of temperature between the up and downcast shafts will suffice to maintain a brisk draught, and that the whole of the heat evolved by the combustion of the fuel must, if possible, be applied directly to heat the column of air in the upcast shaft. The pumping shaft should be generally used, if possible, for the downcast air current, on account of the great absorption of heat that would take place were the furnace placed at its foot. On account of the increase of the surface temperature, and therefore of that of the downcast current, a brisker fire will be required in summer and the reverse will take place in winter. When the pit is not fiery, and the upcast is used solely as an air shaft, and no drawing takes place in it, the furnace may be placed in the main return air-way, which may be enlarged and terminate in the furnace: if the road is through rock, or non-combustible strata, a single arch of brick will suffice; but if the returns are in the coal measures, great care must be taken to prevent, not only the ignition of the coal through the extreme heat, either directly or by spontaneous combustion, but also the expansion of the gas in the seam, and the development of a blower at the very spot where its presence is most dangerous. In this latter case two concentric arches with a space of at least 5 feet must be erected; the space may be well packed with sand. The upper arch will support the rock, and the lower one the bed and grate of the furnace.

The general construction of a furnace is a very simple matter, if the principles and precautions inculcated here form the basis on which the design is prepared. In general practice, the formula

$$H = D \frac{T - t}{T + 459}$$

which gives the *head* of air column, will be sufficient to enable the engineer to calculate the dimensions of the furnace, and, in laying down the ventilation of a colliery, the following are the steps to be taken which are explained in the following chapters.

In practice each foot of area of grate will yield 6,500 cubic feet of air. The amount of air set in motion will be as the square root of the difference of the temperature of the downcast and upcast, and as the square root of the depth from the surface. A furnace 6 feet long by 5 feet broad will give 30,000 cubic feet per minute.

The size, length of the air-ways, their number, the nature of the coals, number of men and horses employed, nature and quantity of goaf (if any), friction, powder smoke, and some other minor details have to be considered first.

The formulæ are to be worked out, and the result arrived at will be that a *given volume of air per second will be found necessary*, whether the ventilation be produced by means of a fan or furnace. This point *must* be arrived at, and neglect or mistakes in calculating it have been the cause of the most disastrous colliery explosions.

The question will then arise, which system will be the best to use, and what are the dimensions required to produce this known quantity of air per second? This chapter has been devoted to the laws affecting the above question, when the upcast furnace is used; the following chapter is devoted to the fan and its general principles.

For all general purposes the following formula will be found useful when furnaces are to be used:—

Add 461 to the temperature of the air in the downcast shaft. This will yield the absolute temperature; the volumes will then vary as the temperature.

To find the velocity of the air current in feet per second for a known head of air, the following will be found useful:—

Let H be the difference of height in each air column or shaft, V the velocity in feet per second.

$$H = \text{head of air in upcast in feet}$$
$$V = 8\sqrt{H}$$

then if the downcast shaft air column is equal to 1,000 feet, and the upcast air column = 1,400

$$400 \text{ ft.} = \text{head of air.}$$
$$H = 400$$
$$8\sqrt{H} = 8\sqrt{400}$$
$$V = 8\ (20).$$
$$V = 160 \text{ ft. per second.}$$

This is given, leaving out the decimal reading ·025, which it is hardly necessary to use in ordinary practice.

CHAPTER II.

THE VACUUM FAN.

Of late years considerable apprehension existed in the minds of engineers respecting the use of a furnace in the pit; danger was always present from gas firing at the furnaces, and, after an explosion, difficulty was frequently experienced in re-lighting the furnaces. M. Guibal proposed to create a circulation of air in the mine by means of the rapid rotation of a fan; this method of creating ventilation has been largely adopted in collieries, and the following data may be taken as indicative of the comparative cost of the fan and furnace systems; but it must be remembered that there are occasions when the furnace is preferable to the fan and *vice versâ*, and that no general rule can be laid down as to which system is to be used; it must be determined on by the engineer, after due deliberation on the circumstances and exigencies of the particular case. The fans now used exhaust the air from the top of the upcast shaft, or at the end of the return air course where they are employed for daylight working. They are divided into two classes: those which keep up a continuous current by centrifugal action, such as Guibal's fan; and those which intermittently discharge a given quantity of air, as the kind of rotary pumps invented by Cooke and Lemielle. One of the main objects of M. Guibal's fan is to have as low a final velocity as possible in discharging the air from the pit; because whatever excess of velocity there may be in the discharged air beyond the velocity of the ascending current is simply waste of power. The fan must be worked with the exact area of discharge opening that is suited to the quantity of air to be discharged. If the discharge opening be too large, there is waste of power, since useless air is put into motion. If the opening be too small, the discharged air meets with a resistance, which also causes waste of power.

At Cannock and Rugeley collieries in one pit, 200 yards deep, the air circulated by the furnace is 103,300 cubic feet per minute; the water-gauge shows 0·86 inches at bank and 0·62 in the pit. The fan (Guibal) gives 180,000 cubic feet at 35 revolutions per minute, with 1½ inches of water-gauge and 8 lbs. of steam pressure. The following tabulated statement shows the comparative cost of fan and furnace under similar conditions:—

Name of Mine.	System.	Coal per H.P. in pounds per hour.	Cost per H.P. in pence per hour.	Coal required per twenty-four hours.	Cost of coal per annum.
					£ s. d.
Tyne Main .	Furnace	43·45 lbs.	·582*d.*	3·67	166 9 6
Hetton . . .	,,	59·60 ,,	0·798*d.*	3·34	152 0 0
Pelton . . .	Fan	12·30 ,,	0·165*d.*	1·03	47 0 0
Elswick . .	,,	10·40 ,,	0·135*d.*	0·87	39 17 6
Whitehaven .	,,	10·70 ,,	0·143*d.*	0·89	40 17 6
Sacré Madame	,,	12·30 ,,	0·165*d.*	1·03	47 0 0

Guibal Fan at Stavely.

Revolutions per minute.	Water-gauge.	Volume of air per minute.	Efficiency of the gross indicated power of the Engine.
	Inches	Cubic feet	Per cent.
32	·70	43,852	40·38
51	1·70	86,283	43·09
64	2·77	101,773	53·27
68	3·10	110,005	53·85

Diameter of Fan = 30 feet × 10 feet wide.

The principal fans for ventilating coal mines in England are the Schiele (Union Engineering Co., Manchester), Guibal and Waddle. For small works the Schiele fan is very useful. Guibal's fan requires a good deal of masonry for its erection, and absorbs a good deal of power. Many mining engineers have a high opinion of Waddle's fan. Objection has been raised against the Schiele fan on account of its having to be driven with a strap: the writer prefers it to the others when carefully attended to and driven by an experienced man in charge.

Subjoined are a series of tabulated results with Schiele's fan of recent construction taken with accurate instruments.

Experiments taken on Schiele's Ventilator at Cortonwood Colliery, Saturday, April 28, 1877.

Steam in boiler	average 60 lbs.
Revolutions of Schiele's fan	,, 162 per min.
Revolutions of engine	,, 64 ,,
Point in stroke where steam is cut off	,, ·25
Average pressure of steam on piston :	
F end	,, 20·0 lbs. per sq. in.
B end	,, 25·3 ,,
Average	22·6 ,,
Effective indicated H.P. of engine	,, 83·4
W.G. in drift door 30 ft. away from fan	,, 1·57 inches
W.G. in pit	,, 1·2 ,,
Quantity of air circulating in mine	,, 219,000 cubic feet
Effective H.P. in air measured in pit	,, 54·0
Per-centage of useful effect, air measured in pit	,, 64·7

Diameter of fan 12 feet. Engine 25″ cylinder, driven by belt from fly-wheel.

Measurements of Air at Hoyland Colliery, Schiele's Ventilator.

July 11, 1878. *Present.*—Mr. J. H. Carrington. *Viewer* (who conducted the measurements).—Mr. Bowling. *Sub-Viewers.*—Mr. J. H. Penman and Mr. Crighton.

Revolutions of fan 175 per minute.

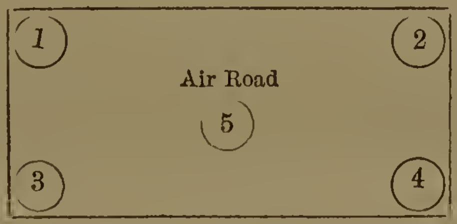
Water-gauge taken 10 yards from fan 1½″.

Places of measurement and quantity of air per minute.

	Speed air meter	Revolutions per minute
Silkstone Seam.		
South Return	1,040	43,680
North Return	1,374	49,464
Parkgate Seam.		
North Return	520	28,210
From North Intake (scaled)	2,740	36,990
South Return	656	36,736
Thorncliffe Seam.		
Main Return	210	14,070
South Return	470	5,757
Floekton Seam.		
North Intake	325	13,975
South Intake	610	30,652
Barnsley Bed (scaled)	300	1,575
Cubic feet per minute		261,109

The closing in of pit top not being in a finished state, there were considerable leakages there, estimated at 10,000 cubic feet per minute, which are not taken into account in the above.

The measurements in every place were taken in five different points in each air road, as under, the mean being taken as the speed of air meter.



The air meter used was a new 6″ improved Biram, by Davis of Derby, and belonging to the colliery.

The fan is 13′ 6″ diameter, and is driven by belt from fly-wheel of engine, 17 feet diameter.

Schiele's Fans,

At the collieries of Sir John Brown and Co., near Rotherham.
(*Copies from Pit Experiment Book of Air Measurements.*)

October 30, 1876.—No. 2, fan 9′ 6″ diameter, on the Barnsley bed.

Engine	98 revolutions per minute.
Fan	236 ,,

W. G. taken at top of upcast $2\frac{1}{2}''$.

Air measured in the Pit.

West horse-road	39,345
East ,, ...	41,270
Old boiler flues	14,940
Scales to stables and heading	8,000
Total cubic feet per minute . .	103,555

Schiele's Ventilator at Harris's Navigation Colliery, Quaker's Yard, Pontypridd. Experiments, February 15, 1880, *and succeeding days.*

Experiment I.—Engine 60 strokes. Fan 159 revolutions. W. G. at fan $1\frac{8}{10}''$.

Fan attached to drift 250 feet area. An opening made in the pit top of 136 feet area. Quantity passing—314,482 cubic feet of air per minute.

Experiment II.—Speed of fan and engine same. W. G. at fan $2\frac{1}{10}''$.

Pit top closed in and fan drawing through pit and through bratticed mouthing between the shafts.

Quantity passing—269,160 cubic feet of air per minute.

Experiment III.—Speed of fan and engine same. W. G. at fan $2\frac{3}{10}''$.

Pit top closed. Separation doors open.

Quantity passing—277,250 cubic feet of air per minute.

Experiment IV.—Speed of fan and engine same. W. G. at fan $3\frac{3}{10}''$

Pit top closed. Most of mouthings in shaft stopped. Fan drawing through pit.

Quantity passing—146,250 cubic feet of air per minute.

In this experiment it is to be noted that the air-road open in the pit for the wind was average 45 feet.

14 feet fan driven by belt from 17 feet diameter fly-wheel on engine.

Experiments on Schiele's Ventilator at Plas Power Colliery, near Wrexham, April 15, 1880.

Steam in engine-house	58 lbs.
Revolutions of fan	129 per min.
Revolutions of engine	129 „
Number of experiments	1
Average pressure of steam on piston:	
F end	14·64 lbs. per sq. in.
B end	16·17 „
Average	15·4 „
Effective indicated H.P. of engine	62
Average number of revolutions of anemometer in south side main return	636·6 per min.
Revolutions of anemometer in north side engine-house scale	63 „
Average number of revolutions of anemometer in north main return	1,093 „
Leakages understated	5,000 feet
Effective H.P. of air in pit	30·2
Watergauge	1·6 inches
Per-centage of useful effect	58
Quantity of air measured in pit	141,920 cubic feet

Fan 13′ 6″ diameter. Engines 20″ diameter cylinder and 20″ stroke.

These data, as far as regards the furnace system, are not accurate as to cost, as the furnaces are of rather antiquated origin, and of late years have been very considerably improved.

Where V = radius of the circle against which the shutters of the fan are fixed.

R = radius of the hollow cylinder of the fan.

M^1 = mass of air contained within the two adjacent shutters $x z$—$z n$, and between the two cylindrical surfaces.

M'' = mass of air contained between the same two shutters, and between the second and third cylindrical surface, at a distance of Z^1 from the centre.

M = mass of air contained between the last two cylindrical surfaces at a distance R from the centre.

Let us suppose the rotation round its centre O of the cylinder to be of a uniform speed V, the volume of air contained within the cylindrical surfaces will be impelled by the shutters, and will be acted upon by the rotary motion of the fan, then the following will denote the centrifugal force developed.

The centrifugal force developed by the mass $M^1 = M^1 V^2 z$.

That developed by the mass $M'' = M'' V^2 z^1$, giving a result for the mass M, $M = M V^2 R$.

In general practice it will suffice to erect a fan guaranteed to give a certain result: the makers of vacuum fans have results which will be sufficiently accurate to go by, and the velocity of the air can be regulated by air regulators; on no account whatever must the regulators be placed in the intake currents, or downcast shaft, as the air being drawn out of the pit at a greater speed than it enters it, a depression will be caused that may produce a blower of gas.

A duplicate engine must be attached ready for use in case of a breakdown, and the engine must be properly governed to run steadily at any speed. Care must also be taken to construct a drift to the fan from the upcast shaft, so that in the event of an explosion the fan itself and its gear may not be damaged. There are so many forms of centrifugal ventilators and vacuum fans, that a description of all would be impossible; the following chapter will give the rules (expressed arithmetically) to find the cubic feet of air per minute required to ventilate the pit, and the fan must be guaranteed by the manufacturer to yield this result with a fair margin, in case of emergency.

Phosphor bronze bearings in the fan itself will be found an economy; and great attention should be paid to the trimming of the oil cups, the firing of the engine, and the air doors in the pit, or the fan is no economy over the furnace.

In designing centrifugal fans the following rules may be observed for proportions. $D =$ diameter of fan, $V =$ velocity in feet per second of the tips of the fan blades. $P =$ pressure in lbs. per square inch.

$$V = \sqrt{p \times 97300}$$

$$P = \frac{V^2}{97300}$$

Length of vane $= \frac{D}{4}$ Diameter of centre $= \frac{D}{2}$

Width $= \frac{D}{4}$ Eccentricity of fan $= \frac{D}{10}$

Note on the Velocity of Air in Mines.

It has been shown that a high velocity is not desirable in the air-ways.

Subjoined is a table of velocities, pressures, and relative power required to effect these pressures and velocities.

Velocity in feet per second.	Relative Pressure.	Relative Power.
3	1·00	1·00
4	1·77	2·37
5	2·77	4·63
6	4·00	8·00
7	5·44	12·70
8	7·11	18·96
9	9·00	27·00
10	11·11	37·03
12	16·00	64·00
14	21·77	101·63
16	28·44	151·70

Thus, if we have an air-way and pass air through it at three feet per second, and suddenly increase that velocity to nine feet a second, we now only require 3 times the pressure to do it, but 27 times the power. The higher the velocity of the air the more coal-dust will be liberated in the mine; and Mr. Galloway's results show that a mixture of coal-dust and air is inflammable if it contains any larger proportion of fire-damp than 0·892 per cent., so that it is a matter of grave consideration for the manager to determine the maximum velocity. Further, a high velocity will blow the incandescent particles of wick through the gauze of every safety-lamp but Williamson's and

Stephenson's, when the miner knocks the top of the wick off his light with the pricker. With Davy lamps the miner should shield his lamp with his body when doing this. The velocities of air in the north of England appear generally greater than in South Wales and Cannock Chase. The reason is that in the north the roofs are bad and the pits not of such recent date, consequently the furnace ventilation yields a velocity that cannot be regulated so easily as with the fan system.

CHAPTER III.

THE LAWS OF CHEMICAL COMBINATION AS APPLIED TO COMBUSTION.

In calculating the velocity and quantity of the air current, the first thing it is necessary to know is the rule for ascertaining the quantity of air required for the complete combustion of the principal form of fuel; and it will be noted that the method of proceeding to arrive at the result must be taken as the rule to follow in all other calculations of a similar nature. The laws of inorganic chemistry will be found to apply here, and a careful perusal of the laws of chemical combination and affinity will enable the reader to arrive at a result with comparatively little labour. In the first place, the combination of any two substances is simply their oxidation with the evolution of heat, and frequently light; such products as are compound products—for instance, coal—cannot be entirely oxidised, *i.e.* their combustion is not complete, and the residue is found in partially consumed particles; this we see in the smoke evolved from a furnace, and it may be taken as a general rule that the more complete the combustion the less the smoke.

There is no destruction of matter or loss of weight in combustion. For instance, a candle is lit and burned out; a small quantity of ash on the end of the wick is all that remains to the naked eye, yet the candle is not destroyed; the ingredients of the candle have combined in certain proportions with the oxygen and aëriform products, viz. aqueous vapour and carbonic anhydride, weighing really *more* than the original candle. This gain in weight represents the exact quantity of oxygen absorbed and used in bringing about the combustion of the candle; and it would be found that the decrease in weight of a volume of air, sufficient to support the entire combustion of the candle, would be exactly equal to the increase of weight of the remaining products of the candle after combustion.

Four laws govern the combination of chemical substances.

I.—When one body combines with another in several proportions, the higher proportions are multiples of the first or lowest. Thus oxygen and hydrogen are contained in water in the following proportion—

16 parts oxygen, 2 hydrogen.

The elements unite to form water. Here the proportion of oxygen is 32 to 2 parts hydrogen.

II.—The same substance always consists of the same elements united in the same proportion.

III.—If two bodies combine with a third body, they are multiples of the proportions in which they may combine with each other. Thus carbon unites with oxygen in the proportion of 12 parts by weight of the former to 16 of the latter. Hydrogen unites with oxygen, as already shown, in the proportion of 2 to 16. Fire-damp is a compound of carbon and hydrogen. Here 12 parts carbon are united with 4, or 2 × 2, parts of hydrogen.

IV.—The combining proportion of a compound is the sum of the combining proportions of its constituents. Thus sulphur peroxide is a combination of sulphur and oxygen in the proportions of 32 to 48, the sum of which is 80; water, of hydrogen and oxygen in the proportion of 2 to 16; the sum of the latter numbers is 18. When sulphur peroxide is added to water, it unites in the proportion of 80 parts of the former to 18 of the latter.

These laws will enable the reader to calculate all questions relating to combustion.

In ventilating a mine, therefore, we have to take into consideration all substances in the mine that will require oxidation, such as the human beings, or rather the human blood, that of the horses, the flames of the lamps, the dilution of the gases, the flame of the furnace (if any), and the gunpowder smoke. These items are the causes that draw on the ventilating current chemically; added to this we shall find that we have causes that affect the volume and velocity of the current mechanically. I propose to deal with these separately. They are both the most important features in ventilation. Neglect of these two features has caused the most disastrous explosions, and the only way to provide against their recurrence is thoroughly to master the scientific laws which govern the former item, and the mechanical and practical causes which affect the latter; having done this, provision must be made for their joint demands, and they both must be provided for, in estimating the velocity and volume of the ventilating current. The means for producing the current must then be determined on; but, firstly, the effective power, whether of the furnace or the fan, as the case may be, must be decided on, on the basis of the formula previously given to arrive at the required result in the two previous chapters.

CHAPTER IV.

CHEMICAL INTERFERENCE WITH THE MAIN AIR CURRENT.

The first and most important is the presence of fire-damp, or carburetted hydrogen.

Its chemical composition is as follows :—

1 atom of carbon to 4 atoms of hydrogen represents marsh gas. CH_4 is, therefore, its chemical symbol. Twice the amount of air renders it violently explosive, and an explosion of fire-damp is thus represented chemically :—

$$\frac{\text{2 vols.}}{CH_4} + \frac{\text{4 vols.}}{2O_2} = \frac{\text{2 vols.}}{CO_2} + \frac{\text{4 vols.}}{2OH_2}$$

The 4 volumes of oxygen which 20 volumes of atmospheric air contain produce 2 volumes of carbonic acid and 4 volumes of steam ; on the condensation of the steam, 16 volumes of nitrogen remain mixed with the carbonic acid. This product is the *after*-damp. Here we have, then, a gas which possesses a great affinity for oxygen, and which, therefore, will weaken the effect of the main air current by absorbing a portion of its oxygen. In what proportion it will be absorbed, volume for volume, may be ascertained by the laws we have previously given affecting chemical combination.

To ascertain *how much air will be polluted by the* *coal gas is the first question*; this is calculated as follows:—The quantity of air discharged into the air-ways of a colliery will always bear a direct relation (i.) to the area of the surface of the coal, such as the roof, side, bottoms, &c., exposed to the action of the ventilating current; (ii.) to the cubic contents of the coal wrought. Each piece of coal wrought discharges through its faces a small amount of gas. We must think of a small lump of coal as if it were a model coal mine.

To estimate the amount of air necessary to dilute the coal gas, we must proceed as follows:—1 cubic inch of coal yields, or is capable of yielding under an air-pump, a volume of gas equal to its own bulk. 1 cubic foot of coal will yield 1 cubic foot of gas.

It will be seen by examining the nature of fire-damp, that unless the gas is diluted with more than 15 times its volume of air, it will be explosive; therefore, 15 × 1 = amount of air required to dilute the gas yielded by 1 cubic foot of coal.

The result is obtained by *multiplying the cubic contents of the coal in feet by* 15. The result is given in cubic feet of air. We advise our readers to increase this number 15 to 100, to provide for contingencies, such as blowers, &c., then 100 cubic feet of air will be required for each cubic foot of coal wrought in the seam.

Next, we have the withdrawal of oxygen from the atmosphere in the pit, consequent on combustion, respiration, candles, lamps, and furnaces.

We have to allow for these as follows :—

If we have a furnace at the foot of the upcast shaft, then, since in oxidation 1 atom of carbon unites with 2 atoms of oxygen, following the laws of affinity in chemistry, we find that the atomic weight, or figure denoting the combining proportion of carbon, is 12, that of oxygen 16.

The oxygen will be represented by 32, since there are 2 atoms of this gas, the carbon by 12.

$32 \div 12$ will give us the amount of oxygen required to oxidise a piece of carbon. Let us suppose then that 1 lb. of coal is to be burnt in the furnace per second. Then $32 \div 12 =$ oxygen in lbs. per second $= 2\frac{2}{3}$ lbs.

1 lb. of hydrogen will require 8 lbs. of oxygen by the same rule, for—

1 being the atomic weight of hydrogen,
16 „ „ „ oxygen,

$$H_2O = \text{the product,}$$

$$\text{or } 16 \div 2 = \text{lbs. of oxygen,}$$

$$= 8 \text{ lbs.}$$

The composition of air inhaled and exhaled is given in the following tables as a reference.

Before entering the lungs :—

Nitrogen	79·0
Oxygen	21·0
	100·0

When exhaled from the lungs:—

Nitrogen	79·0
Oxygen	17·5
Carbonic Acid	3·5
	100·0

3½ units of oxygen are, therefore, extracted from the air by the action of the lungs, and applied to oxidise the carbon in the blood cells. A cubic foot of air breathed will be, therefore, vitiated by 3½ per cent. of carbonic acid gas; therefore, each person will discharge from his lungs into the mine 0·035 × 60 = 2·10 cubic feet per hour.

Medical authorities state that an average inhalation of air is represented by 1 cubic foot per minute when the person is awake.

A horse breathes 6·3 times the volume per minute of a man.

$$\cdot 035 \times 6\cdot 3 \times 60 = 13\cdot 230 \left\{ \begin{array}{l} \text{cubic feet of carbonic acid} \\ \text{per horse per hour dis-} \\ \text{charged into the mine.} \end{array} \right.$$

We have now to consider the effect of the lamp and candles, and the gunpowder. We shall then be in a position to estimate the total *chemical vitiation of the air of the mine for a given time.*

A candle or lamp produces about 2·51 cubic feet of carbonic acid per hour by experiment; this is shown by the fact that if a candle is burned in a close vessel containing 1 cubic foot of air, ·20 of the volume of air is deprived of the oxygen, since it is reduced in bulk to that, or nearly that, extent. The proportion of oxygen in air is 21 per cent., therefore,

$$\cdot 20 \times \cdot 21 \times 60$$

$$= \text{cubic feet vitiated per hour per light,}$$

$$= 2\cdot 52.$$

I find, by experiment, that 2·51 is nearer the mark as an average reading. Let x = number of lamps used, then $2\cdot 51 \times x$ = air in cubic feet per hour vitiated by lamps, the rule being to multiply the number of lamps used by 2·51.

Gunpowder is our next source of vitiation, and is a most important one, since the presence of nitrous fumes proves so very detrimental to the health and the eyesight of men and horses in a mine. M. André estimates the effect of ordinary blasting powder to be as follows:—

1 lb. of powder produces 0·30 lb. carbonic acid. 1 cubic foot of carbonic acid weighs ·1164 lb. per pound of powder consumed.

$$\frac{\cdot 3000}{\cdot 1164} = \text{cubic feet of carbonic acid gas discharged}$$

per pound of powder fired.

$= 2\cdot 58$ cubic feet per lb. of powder.

White-damp, or carbonic oxide, is another source of evil, and is very often ignored by engineers.

Its affinity for oxygen is very great. It cannot be found with a candle; and here let me take the opportunity of warning well-sinkers that to test the atmosphere of the shaft with a candle is the most foolish and wrong proceeding; by all means test for the presence of carbonic acid gas with a candle, but it must be borne in mind that carbonic oxide is a *poison*; it does not suffocate, but actually kills men by its poisonous action on the blood, producing fainting and giddiness, and because a candle burns in a well it should never be assumed that the atmosphere is sufficiently pure for a man to live in it. An atmosphere of one per cent. of this gas is fatal to warm-blooded animals. It is lighter than air, and will therefore be found in the roof, and will be detected by its peculiar odour; imperfect spontaneous combustion of wood and coal produces it; small explosions of fire-damp in a large space produce it also. Fortunately, in estimating the volume of the air current, one may neglect to take into account the presence of carbonic oxide, as if precaution be taken it will not present itself.

We have now got possession of all the data to estimate the *total* amount of chemical vitiation as follows:—

Fire-damp allowed for as under:

1 cubic foot of coal requires 100 cubic feet of air.

Furnace; 1 lb. of coal requires $2\frac{2}{3}$ bulk of oxygen per second.

1 horse discharges 13·230 cubic feet of carbonic acid per hour.
1 man ,, 2·10 ,, ,,

1 lamp discharges 2·51 cubic feet of carbonic acid gas per hour.
1 lb. of powder „ 2·58 „ „

Added to this, we must allow for effluvia 2 cubic feet of air per minute per person.

2·10 cubic feet of carbonic acid, discharged per hour per man, must be diluted to allow the air in the pit to contain not more than ·35 per cent. of carbonic acid, or the men would be gradually choked off. This will require 600 cubic feet of air. Each man requires 600 cubic feet of air per hour = 10 cubic feet per minute.

Then each lamp will require, by proportion, 720 cubic feet per hour, or 12 cubic feet per minute.

If each man has a lamp,

$$10 + 12 = \left\{\begin{array}{l}\text{cubic feet of air per minute}\\ \text{for the man complete.}\end{array}\right.$$

10 + 12 + 2 = the amount, allowing 2 cubic feet for effluvia.

= 24 *cubic feet per minute per man and lamp employed.*

Horses :—

Each horse produces 6·3 times more the amount of carbonic acid than a man.

As a general rule, 3 men equal 1 horse in air breathed.

∴ 3 × 24 = cubic feet per horse per minute.
= 72 cubic feet per minute per horse employed.

Mr. André, in his able work on mining engineering, states that 'the quantity of air required by eight men will be taken up by consuming 1 lb. of powder,' for reasons set forth previously.

Assuming this to be correct, which may certainly be done,

$8 \times 24 =$ cubic feet of air per lb. of powder fired.
$=$ 192 cubic feet per lb. of powder fired.

In estimating the amount of air required for dilution as a factor of safety, 2, or twice the amount required, is just within the limits; when blowers exist, from 2 to 6 times the amount must be allowed.

Now the formula adopted by nearly all mining authorities to estimate the quantity of air required in cubic feet per minute per district, if the mine is divided off into districts, is

$Q =$ quantity of air per minute in cubic feet.
$M =$ number of men at work in the mine.
$H =$,, horses ,, ,,
$P =$ lbs. of gunpowder fired per hour.
$O =$ output, or quantity of coal raised per minute.
$A =$ area of surface coal exposed to the ventilating current, in yards.

Then $Q = M\,24 + H\,72 + P\,192 + O\,100 + A$

Where 1 ton of coal wrought = 1 cubic yard of coal.

Then if the output be 100 tons, the expression O = 100 cubic yards.

The average quantity of coal raised per minute, or output, is thus calculated.

If the total output raised is 300 tons in 12 hours by, let us say, 60 men,

Each man raises 5 tons in 12 hours.

$$720 : 1 :: 5$$

or $\frac{5}{720}$ of a ton per man per minute.

Then in a mine where there are 400 men, 30 horses, and the coal surface is 1,000 yards,

The output 600 tons per day of 12 hours, and the powder used 8 lbs. per hour,

$Q = 24\,(400) + 72\,(30) + 192\,(8) + 100\,(17\tfrac{2}{3}) + 1000.$

$Q = 9600 + 2160 + 1536 + 1766\tfrac{2}{3} + 1000.$

$Q = 16062\tfrac{2}{3}$ cubic feet per minute.

CHAPTER V.

MECHANICAL INTERFERENCES WITH VELOCITY OF THE AIR CURRENT.

We have seen in the previous chapter that allowances must be made for the consumption of oxygen, vitiation of the remaining air, and the dilution of the gases, in estimating the *quantity* of the air current.

The *velocity* of the current will be affected by three things: *the friction of the air along the sides of the air-ways; the divisions of the main air current; and the position and extent of the stoppings and air doors.*

The friction of the air current will depend on the *area* of the rubbing surface traversed by the air; in the case of fluids, the friction is overcome by increasing the head, and the fall is thus calculated.

L = Length of channel tube in yards.

A = Cross sectional area of the stream in square feet.

P = Perimeter in feet.

F = Fall or difference of level in inches of the two ends of the pipe.

C = Cubic feet of water discharged per minute.

$$F = \frac{\left(\frac{C}{A}\right)^2 \times L \times P}{874320 \times A}$$

We cannot, however, make a 'fall' for the air current, but we can increase the 'head;' and the velocity of the air current must be sufficient, not only to overcome the friction, but to sweep the face of the seam sufficiently fast to secure good ventilation; a large volume of air at a medium velocity is better and safer ventilation for a fiery pit than a small quantity at a high velocity. The whole pit must be filled with *air*, and this can only be accomplished by a larger volume, if possible, than it will hold. At Haswell Colliery, $\frac{10}{11}$ of the ventilating pressure is spent in overcoming the friction of the air; the remaining $\frac{1}{11}$ creating the final velocity at the top of the upcast shaft. The amount of friction of the air can only be estimated by the amount of force required to overcome that friction; but since the friction is directly affected by the area of the rubbing surface, the pressure required to overcome the friction will be proportionate to the area of the rubbing surface; and if this area is multiplied by any number, the friction, and consequently the pressure required to overcome it, is increased in a like degree. To get the greatest effect with the least amount of friction, we require to find a figure whose perimeter or circumference will be least for the greatest sectional area.

The perimeter of a square is 4 when its sectional area is 1, but the perimeter of a circle is 3·1416 for an equal area of 1.

Then we should, if possible, use circular air-ways;

the friction of any fluid, whether air, gas, or water, varies as the density of air, gas, or water.

An air-way 4 ft. × 5 ft. has a perimeter of 20 feet; the perimeter multiplied by the length gives the area or rubbing surface r.

$$r = p \times L$$

Where r is the area, p the perimeter, and L the length.

So that if the above air-way were 1,500 feet long, the rubbing surface would be 30,000 square feet; in an air-way 4 ft. × 10 ft. and of an equal length the rubbing surface $= 4 \times 10 \times 1{,}500$.

$$= 60{,}000 \text{ square feet.}$$

Thus for *four* times the area there is only *twice* the extent of rubbing surface; if we divide this air-way into four equally small ones, such small air-ways will have a perimeter of 20 feet; therefore the total perimeter of the four air-ways is equal to $4 \times 20 = 80$ feet.

$$\begin{aligned} 80 \times 1{,}500 &= \text{total rubbing surface.} \\ &= 120{,}000 \text{ square feet,} \end{aligned}$$

the friction being double.

Therefore one large air-way is more economical than a number of small ones making up the same size as the large air-way, but frequently it is more convenient to have a number of small ways than one large one. A feature to be recollected in speaking of the pressure employed in ventilating a colliery is that it is the pressure in pounds per square foot that is meant, and that the pressure will depend naturally on the area over which it is distributed; for instance, 10 lbs. per square foot on an air-way containing 100 square feet is a *total pressure* of 1,000 lbs.; in mining the pressure is reckoned in lbs. per square foot of sectional area of the air-way, as in mechanical engineering the total force acting upon the piston of a locomotive engine is obtained by multiplying the pressure per each square inch into the area of the piston.

The resistance to the air depends on the perimeter of the way, and the pressure per square foot necessary to overcome the friction will vary as the sectional area of the air-way, as in the steam-engine.

The laws governing the friction of fluids in pipes are as follows:—

(i.) For a given section of an air-way, the resistance from friction will increase directly as the length of the airway.

Thus, an air-way 5 ft. × 4 ft. has a perimeter of 20 ft.; if the way is 10 ft. long the rubbing surface is 20 × 10, or 200 square feet; if the air-way is increased in length to 20 ft., the rubbing surface is 20 × 20, or 400 square feet; if we double the air-way in length, we double the rubbing surface, and *vice versâ*. This feature points out the necessity of making each air-way of nearly equal length, and the 'run,' *i.e.* the distance the air has to travel, should be as short as possible.

(ii.) The resistance due to friction will vary as the area of section varies.

The perimeter of a circle was shown to be 3·1416 when its diameter is 1; that of a square is 4 when its diameter is 1. The perimeter of any rectangular figure is obviously greater than that of a circular figure.

We cannot use circular ways, and, therefore, square ones are the next best.

(iii.) The resistance due to friction in any air-way of a given sectional area is less than the resistance due to friction in several air-ways of a respectively less area, but the sum of whose area is equal to the area of the single air-way.

Thus, in an air-way 10 ft. × 10 ft. the perimeter is 40 ft.

Let us suppose two air-ways, each half the size of the large one, the diameter of which is 40 ft.; then we have two ways, 5 ft. × 5 ft., whose perimeters each equal 20 ft.; their total perimeter equals 40 ft., but their respective areas being half the large one, four of the areas of the small ways will equal the area of the large one. The friction will be twice the amount in the large way for the two small ones; therefore, one large air-way, as before explained, is preferable to two small ones.

The friction of the air has, therefore, to be overcome by pressure on the air column, read and estimated in lbs. per square foot. This pressure, we have seen, will vary with the area, as in the case of the steam-engine.

If we have two air-ways, one 5 ft. × 4 ft., the other 4 ft. × 3 ft. in section, the area of the first is 20 ft., and the second 12 ft. $\frac{12}{20}$ of the pressure required by the former will serve for the ventilation of the latter. If we treble the velocity of the air current, we treble the extent of the rubbing surface, since three times the amount of air passes over its surface in the same space of time that one volume did. This is the foundation of the assumption that *the resistance increases as the square of the velocity*, and this is the law that governs friction under variable velocities.

A previous chapter showed how to find H, the *head of air* necessary to produce ventilation. The head being expressed in feet, multiply the head by 0·0765 lb. (pressure of 1 ft. of air at 60° F.), and divide the product by 5·2 lbs. (pressure of 1 in. of water), and the result is the watergauge reading.

The head (H) is a fraction (K) of the product of the rubbing surface (S), multiplied by the square of the velocity (V) in feet per second, divided by the sectional area (A) of the air-way.

Then—

$$H = \frac{K\,S\,V^2}{A}$$

The value of K has been determined as for a velocity of one fraction per second.

$K = 0{\cdot}00011.$

$K = 0{\cdot}00033$ for a mine,

and ·00066 will be found more reliable in practice.

Thus, to find the resistance of an air-way 7 ft. × 8 ft., 1,000 ft. long, the velocity (V) being 4 ft. per second.

$S = 60{,}000$ square feet.

$a = 8 \times 7 = 56$ square feet.

Then— $h = \dfrac{{\cdot}00066 \times 60{,}000 \times (4)^2}{56}$

$$h = \frac{{\cdot}00066 \times 60{,}000 \times 16}{56}$$

$= 11{\cdot}31$ ft. of pressure or

$0{\cdot}17$ in. of water-gauge.

The following formulæ are now all that are required to be known.

Let P = pressure per square foot.

A = sectional area in square feet.

S = rubbing surface in square feet.

V = velocity in 1,000 feet per minute, as a unit of velocity.

K = co-efficient of friction.

To find total pressure

$$Pa = K\,S\,V^2$$

Rubbing surface $S = \dfrac{Pa}{K\,V^2}$

Velocity squared $V^2 = \dfrac{Pa}{K\,S}$

Co-efficient of friction $K = \frac{Pa}{S V^2}$

This may be always taken as ·0217 cubic foot per minute (constant)

Pressure per foot $P = \frac{K S V^2}{a}$

Volume = area velocity

Area of section $a = \frac{K S V^2}{P}$

To find total pressure

$$a \times \frac{K S V^2}{a}$$

The quantity of air circulating in a mine varies as the square root of the pressure employed to circulate it; the ventilative effect, if we may so term it, increases with the square root of the height of the motive column.

The application of these formulæ will show the reader that *quantity per minute*, and *not velocity*, is the essential qualification of safe and economical ventilation; for it has been seen that the higher the velocity the greater the friction, that this increase is not in direct ratio, but as the square of the velocity; and therefore to obtain this result we must *split* or divide the main air current. The following general observations on splitting the main air current may be found useful. In the first place it reduces the velocity of the air current, and therefore the friction; but, on the other hand, the increased quantity and velocity of the air in the up and downcast shafts produces increased friction, and this, if carried to excess, effectually limits the advantages which can be derived from splits. The splits can be made in a limited quantity only, or the velocity of the air current will be too feeble; on the other hand, if the splits are too few, the friction, consequent on a high velocity, will be too great. The exact mean must be determined. For air velocities the following may be taken as a reliable rule, except in very exceptional cases: 5 feet per second is the limit of safety for the velocity of any part of the air current of a mine where non-extinguishing lamps, such as Davy's or Clanny's, are used; 2 feet, on the other hand, is too slack a current to blow away powder smoke and sweep the sides, roof, and face free of gas: the average speed is 3½ feet per second. With a number of splits with an equal amount of air allotted to each, the shorter splits must be 'stopped' or obstructed, artificially, to make their frictional resistance, after allowing them their proper share of air, equal to that of the largest splits, and these stoppings obviously lessen the total quantity of air in circulation.

Every split should begin as near the bottom of the downcast shaft as possible, and should empty into the main return air-way as near the upcast shaft as possible. In all cases regulators are to be avoided if possible, and this can be nearly always secured by making the splits of an equal length.

Stoppings or barriers should be made air-tight, and strong enough not to be blown out in the event of an explosion. Two curved brick walls, with their convex side outwards, allowing 5 feet space, which should be filled with sand, make a good sound stopping; and, in the event of an underground fire, are fairly fireproof. In an air-way of a given section it is obvious that no more air, as regards quantity, can pass through the air-way, in a given unit of time, than the air-way will hold, and that therefore the velocity of the air to a great extent governs its quantity, but the greater the velocity the greater will be the friction, and here lies the fundamental element of success in all colliery ventilation.

Mr. Atkinson showed that if a mine had such shafts and air-ways that there were five equal splits of air, the resistance of the shafts equalled half those offered by the air-ways; then if the air was not split at all and the velocity was 10,000 cubic feet per minute, the following was the result obtained by splitting :—

No. of Currents or Splits	Total air velocity per minute	Quantity in each Split
	Cubic feet	Cubic feet
1	10,000	10,000
2	27,892	13,946
3	49,449	16,480
4	71,527	17,882
5	90,789	18,150
6	107,800	17,966
10	141,714	14,171

And the coal burnt in the upcast furnace, or in the engine driving the vacuum fan, would increase in the same proportion with the quantity of air, and the power absorbed would increase in that ratio ; otherwise, if the power and coal consumed remained the same for 10 splits as for 1, we should have for one airway 10,000 cubic feet of air per minute, and for 10 splits only 58,556. One of the worst features in the careless laying out of a mine consists in working the pillars away too near the shafts at first starting ; no room is left to make splits at a future period, without incurring some risk and great expense.

When the stoppings to be erected are situated in a road used for travelling, they must be replaced by air doors.

The position and construction of these doors is of the very greatest importance. A badly fitting or warped door may cause explosion, and the existence of air sheets in place of doors may allow the explosion to spread and fire the whole pit. The rules to be observed are—that every door shall remain in its proper position by being properly weighted ; that if to be left open, the word 'Open' should be plainly painted on both faces of the door, or, if to be left shut, the contrary. If air sheets are used, they must be made of non-inflammable brattice cloth, such as Scott's, and *not of tarred sacking.* When naked lights are used, they should not be stuck on the sides of the air doors ; the worst fire the writer ever saw in a colliery was due to this practice. If wooden doors are used, they must be thus constructed:—They should be made of oak planks, $\frac{5}{8}$ in. thick, tongued to prevent leakage at the joints, and felted over the space between the door and the lintel and posts to reduce leakage. The posts should be well set back in the side of the way in brickwork, and the doors should open the way the tubs generally travel; if there is much traffic through the door, it answers to place a boy there to 'trap' the door, *i.e.* to open and close it after each train of tubs has passed, as the loss of air will show a considerable increase of expense in the ventilating charges. It will be found useful to keep a sheet of these charges, in order to show the cost of the air current for a given quantity of coal raised. A safety door should always be added to the ordinary doors in important places. This door consists of a door lying on the roof, and kept up by a catch; any violent explosion blows away the catch, and, in so doing, liberates the door, which falls into the position occupied by the ordinary air door; the spread of an explosion and the deadly effects of after-damp are thus guarded against, and the arrangement cannot fail to recommend itself to colliery engineers for its simplicity and cheapness. It must be continually borne in mind that in descending the pit, the miner or engineer is in a position where his life and those of his fellow-men depend on his individual care, and that explosions are the certain consequences of any carelessness that is tolerated to exist as an unchecked practice; and the careless way in which air doors are left open in some collieries by miners is the most eloquent argument that they should be weighted, by which means a most fruitful source of danger and annoyance to the other men in the pit by the interruption of the volume of ventilating current would be avoided.

However carefully the resistance of splits may be calculated, trams, falls of roofs, and pit-props left in the roads will upset the results then arrived at, so that we only give the mathematical means by which the results may be estimated, so as to form an opinion of what is likely to be the *actual* result of splitting in a particular way.

We think the tendency of modern mining works on ventilation is rather too much in a mathematical direction.

Air Splits.

In mining ventilation these are divided into two classes, equal and unequal splits. Equal splits are thus formed. An air-way of a given length is divided into two air-ways, which will be of the same area as the original air-way. We will take as an example an air-way with 40 feet area; it is of such a length that the rubbing surface is 200,000 sq. feet. If we split this air-way equally we shall get an area of 40 feet in each and only 100,000 sq. feet of rubbing surface; if we split it into five equal splits we shall get five times the area, 40,000 sq. feet, of rubbing surface. From these considerations we deduce the rule that the rubbing surface will remain unaltered in area when equal splits are made, although the area of those splits will be increased.

But it will be seen that, inasmuch as no more air can pass through an air-way in a given unit of time than the area of the air-way will permit, so also will the area, or diameter, of the shafts influence the *effect* of splitting by the resistance due to friction against their sides, caused by the increased quantity of air passing down them. Let us take the following example.

Perhaps the best illustration of splitting is Mr. Wardle's.

An air-way discharges 30,000 cubic feet per minute, it is required to find the total discharge on adding a similar air-way.

Secondly, the power being constant, each air-way being 5 ft. × 10 ft. × 1000 ft. long, required the total discharge.

The first problem is thus treated:

The dimensions of the air-way are 10 ft. wide × 5 ft. high.

The area will be the width × height = 50 sq. ft.

The perimeter will be twice the width
added to twice the height . . = 30 ft.

The length of airway was shown . = 1,000 ft.
The rubbing surface will be the length
× the perimeter, 1,000 × 30 . . = 30,000 ft.

Coefficient of friction (constant) ·0217 cubic ft. per min.

$$\text{Velocity} = \frac{\text{volume}}{\text{area}}$$

$$\frac{30{,}000}{50} = 600 \text{ ft. per min.}$$

Total pressure = coefficient of friction × rubbing surface × velocity squared.

Total pressure = $\cdot0217 \times 30000 \times 6^2 = 234{\cdot}36$.

Pressure per square foot is the total pressure divided by the area

$$\text{or } \frac{234{\cdot}36}{50} = 4{\cdot}6872 \text{ lbs. per square foot.}$$

Note.—When the volume is unknown the velocity is thus calculated.

$$\text{Velocity squared} = \frac{Pa}{KS}$$

$$\text{in this case} = \frac{234{\cdot}36}{\cdot0217 \times 30000} = \cdot36$$

$$\sqrt{\cdot36} = \cdot6$$

So far we have found the total pressure to be 234·36 lbs. per square foot.

We now require to find the volume (total) after adding the additional air-way as set forth in the second part of the problem.

The first air-way was given to be 10 feet × 5 feet = 50 feet area in section.

50 × 2 = sectional area of the two air-ways = 100 square feet.

The velocity must first be found.

By the previous formula

$$V^2 = \frac{\text{Total pressure}}{\text{Coefficient of friction} \times \text{rubbing surface}}$$

$$V^2 = \frac{234{\cdot}36}{{\cdot}0217 \times 60000}$$

$$V^2 = \frac{234{\cdot}36}{13{\cdot}02}$$

then $V^2 = {\cdot}18$

$$V = \sqrt{{\cdot}18}$$

$= {\cdot}42426$ in thousands of feet per minute, or 424·26 cubic feet per minute.

The volume will now be found by multiplying the area into the velocity, thus,

$$100 \times 424{\cdot}26$$

= 42426 cubic feet total discharge.

The pressure required to pass a volume of air through a large passage as compared with a small is in inverse proportion to the square of their areas.

The velocity per minute is the square of the area of the air-way divided by the volume in cubic feet per minute.

To find the dimensions of an air-way capable of passing a known quantity of air per minute at a known velocity, the volume, divided by the velocity in feet per minute, must first be arrived at. The square root of this result multiplied by four will give the perimeter of the air-way required.

To find the horse-power absorbed or expended to pass a known quantity of air at a known velocity through an air-way of known size and length. The perimeter multiplied by the length will give the rubbing surface; the velocity in feet per minute should be divided by 1000 to reduce it to thousands of feet per minute; this result should be squared, and then multiplied by the coefficient of friction. This result should be multiplied into the rubbing surface in square feet and by the constant ·080728 lb.

This last result should be multiplied into the volume in cubic feet per minute. This yields the number of foot pounds absorbed in producing the work. Now 33,000 foot lbs. represent 1 English horse-power; therefore, if we divide the last result obtained by 33,000, we shall have the horse-power (or by 32,549 for French horse-power).

If a known quantity of air passing a known depression or watergauge reading is obtained, and it is required to know what quantity is passing when a different watergauge reading is registered, the following rule must be proceeded on.

The quantity of air passing will be the square root of the different watergauge reading obtained, multiplied into the known quantity of air previously passing, divided by the square root of the watergauge reading obtained with that known quantity of air, or,

A = Watergauge reading with B quantity of air.
B = Known quantity of air in cubic feet per minute, passing to give A watergauge reading.
Q = Quantity of air passing (to be found) with W watergauge reading.
W = Watergauge reading when quantity of air passing per minute is required to be found.

$$Q = \frac{\sqrt{W} \times B}{\sqrt{A}}$$

Inversely this proposition stands thus. To find the height of the watergauge when both quantities of air are known and only one watergauge reading. Thus, suppose

$A = \cdot 72$
B = 25000 cubic feet per minute.
Q = 75000 „ „
W is required to be found.

Then the square of the quantity is directly proportional to the reading of the watergauge. Thus the square of the watergauge reading will equal the quantity passing.

Or as 25,000 is one-third of the amount 75,000 three times the original watergauge reading squared.

= watergauge reading required. Thus

$3^2 = 9$

$9 \times {\cdot}72 = 6{\cdot}48$ inches of watergauge

$W = 6{\cdot}48$ inches.

To double the quantity of air the pressure must be increased as the square of the quantity; thus if a pressure of 3 lbs. yields 10,000 cubic feet per minute, and we require 20,000 cubic feet,

3^2 = pressure required, or 9 lbs. pressure per square foot.

The above examples and remarks are intended to simplify, if possible, the theory of air splits. It will be found, however, that in practice the result will never equal the expectation of the engineer derived from his theoretical calculations. This is owing to friction in the shafts, irregular air-ways in perimeter, trams standing on the roads, &c. For reference to Belgian Mining Works the following table is inserted :—

1 kilogrammetre = 7·23314 foot pounds.
1 foot pound = ·13825 kilogrammetre
1 force de cheval = 4500 kilogrammetres per minute
,, ,, = 32549 foot pounds
,, ,, = ·98757 English horse-power
1 English H.P. = 1·0319 French ditto.
1 lb. per square in. = ·0703 kilogrammes per square centimetre
1 kilogramme per square centimetre = 14·22 lbs. per square in.

For table to compare lbs. per square inch, and kilos. per square centimetre, refer to Molesworth's 'Pocket-book of Engineering Formulæ,' p. 405.

CHAPTER VI.

INSTRUMENTS USED IN VENTILATION.

These are the watergauge, barometer, thermometer, and anemometer.

Mr. Daniel considers that the watergauge is a sufficient indication of the resistance encountered by the air current in the air-ways of a mine, since the height of the gauge increases as the length of the ways are inversely as their sectional area.

Every important air-way in a mine must have its watergauge, and we strongly recommend that those men who have to read the gauge should be examined from time to time on the subject.

Principle of the Watergauge.

The watergauge is an instrument to indicate to the eye the force of the air current. By its use the loss occasioned by resistance due to friction of the air in motion through the air-ways can be calculated.

The instrument consists of a U-shaped tube of glass, and in the bend lies a small quantity of water.

The top of one leg of the tube is left open, that of the other terminates in a pipe communicating with the upcast shaft by being simply turned into the dumb-drift. Thus the instrument becomes a model of the mine, the right leg of the instrument being open to the ordinary atmospheric pressure representing the downcast shaft, the leg communicating with the dumb-drift the upcast shaft. The vacuum produced by the rapid rotation of the fan sucks the air out of the drift, and consequently out of one leg of the watergauge; the column of water in the right leg of the instrument is heavier than that of the column in the left leg. Consequently the former presses down the latter until they balance each other; this is arrived at when the amount of water that the difference of level in the two columns consists of is equal in weight to the amount of exhaust, and consequently weight of air in the upcast shaft. The gauge is then read by screwing down the vernier till the line marked *o* is level with the top of the column of water in the right-hand or open leg of the gauge; the figure or reading level with the top of the column of water in the left leg is the 'watergauge reading,' or head of water.

The greater the friction in the air-ways, the higher will be the watergauge reading; thus a fall of roof is usually followed by an increase of watergauge.

The important formulæ and consequent information that can be gained from the watergauge are as follows :—

To find length of motive column in feet.

$\frac{\text{Pressure in lbs. per square foot}}{\cdot 0763122}$ = length of motive column in feet.

To convert exhaust reading by aneroid or barometer to inches of watergauge.

1 inch of water = ·036125 lb. per square inch

1 ,, mercury = ·4897 lb. per square inch at 62° Fahr.

Then 1 inch of mercury = $\frac{\cdot 4897}{\cdot 03612}$ inch of water

Or 1 ,, ,, = 13·56 inches of water

Or $\frac{1}{10}$,, ,, = 1·356 ,,

So that to convert inches of mercury to inches of water—

Let M = inches of mercury in tenths of an inch.

Let W = inches of water ,, ,,

$W = M \times 1{\cdot}356.$

On experiment I tried to compare the accuracy of readings by my patent aneroid apparatus, and the watergauge showed the following, when the readings were reduced to pressures in ounces per square inch.

Watergauge reading = 2·2 inches = $1\frac{167}{615}$ ounces per square inch.

Difference of atmospheric reading of barometer and that of patent apparatus, *i.e.* exhaust reading. $\Big\}$ $= 1{\cdot}3$ inch $= 1\,\frac{1261}{625000}$ oz. per sq. in.

The difference of the watergauge reading due to friction and the whole watergauge reading gives the pressure that creates the final or theoretical velocity of the air in the upcast shaft; this averages ·20 inch.

To arrive at this

t = temperature of air in upcast.
V = velocity in feet per second.
M = motive column of air in feet.
W = watergauge reading reduced to decimals of an inch.

$$V = 8{\cdot}025\sqrt{M}$$

$$M = \frac{V^2}{64{\cdot}4}$$

$$W = \frac{\dfrac{1{\cdot}32529 \times 30}{459 + t} \times M}{5{\cdot}196}$$

Thus W represents inches of watergauge in decimals absolutely used to create the final velocity in the upcast shaft; this, subtracted from the whole watergauge reading, gives as the remainder the watergauge reading, and consequently the pressure used in overcoming the friction due to the sides of the air-ways.

Thus with a fall of roof the final velocity in the upcast shaft may remain very nearly the same as before it occurred, but the watergauge may increase half an inch in height, or more. If there were no friction in a mine, and the area of the air-ways were equal to the area of the shafts, there would be no watergauge reading at all, assuming it possible that such a theoretical equilibrium between the air columns in the shafts could be secured. From this, therefore, we lay down the rule that the manager (*vide* 'Coal Mines Regulation Act,' 1872) should take the final velocity in the upcast shaft with an anemometer, note the watergauge, and calculate out by the above formula what pressure is absorbed by friction in the air-ways, and what is left over to produce the *theoretical* final velocity in the upcast shaft.

The following formula will be required to compare the height of watergauge reading and that of a motive column of air in feet, and to arrive at the velocity in feet per second due to this column.

W = watergauge reading in inches.
P = pressure in pounds per square foot.
M = motive column of air in feet.
V = velocity in feet per second of air caused by M.

$$W = \frac{P}{5{\cdot}196}$$

$$P = \frac{W \times 62{\cdot}355}{12}$$

$$M = \frac{P}{{\cdot}0763122}$$

$$V = 9{\cdot}025 \sqrt{M}$$

It is impossible to insist too strongly on this point of separating the watergauge reading due to *resistance from friction* and that due to the final theoretical velocity from the total watergauge reading.

The following represents the form of watergauge book for fan-engine-room, that should be kept for the manager.

Barometer	Engine stroke per minute	Watergauge in inches	Air velocity in upcast shaft by anemometer	Temperature of air in upcast shaft
1. . . .				
2. . . .				
3. . . .				
4. . . .				

From —— to —— signed —— fan-engine-driver on duty —— relieved at——

This should have four entries in the engine-driver's 8 hours shift, and should be sent to the manager's office. There will be three books for 24 hours on an 8 hours shift. This gives ample time to calculate the two results above insisted on that should be entered in a book there.

Date	Barometer	Total watergauge	Proportion due to friction	Proportion due to velocity (final)

Barometric Pressure Table.

Gauge and Barometer in inches	Pressure per sq. in.	An exhaust of	
	lbs.	lbs.	
26·00	13	2	
26·1	13·05	$1\frac{19}{20}$	
26·2	13·1	$1\frac{9}{10}$	
26·3	13·15	$1\frac{17}{20}$	
26·4	13·2	$1\frac{4}{5}$	
26·5	13·25	$1\frac{3}{4}$	
26·6	13·3	$1\frac{7}{10}$	
26·7	13·35	$1\frac{13}{20}$	
26·8	13·4	$1\frac{3}{5}$	
26·9	13·45	$1\frac{11}{20}$	
27·00	13·5	$1\frac{1}{2}$	
27·1	13·55	$1\frac{9}{20}$	
27·2	13·6	$1\frac{2}{5}$	
27·3	13·65	$1\frac{7}{20}$	
27·4	13·7	$1\frac{3}{10}$	
27·5	13·75	$1\frac{1}{4}$	
27·6	13·8	$1\frac{1}{5}$	
27·7	13·85	$1\frac{3}{20}$	
27·8	13·9	$1\frac{1}{10}$	
27·9	13·95	$1\frac{1}{20}$	
28·00	14	1	oz.
28·1	14·05	$\frac{19}{20}$	$=15\frac{1}{5}$
28·2	14·1	$\frac{9}{10}$	$=14\frac{2}{5}$
28·3	14·15	$\frac{17}{20}$	$=13\frac{3}{5}$
28·4	14·2	$\frac{4}{5}$	$=12\frac{4}{5}$
28·5	14·25	$\frac{3}{4}$	$=12$
28·6	14·3	$\frac{7}{10}$	$=11\frac{1}{5}$
28·7	14·35	$\frac{13}{20}$	$=10\frac{2}{5}$
28·8	14·4	$\frac{3}{5}$	$=9\frac{3}{5}$
28·9	14·45	$\frac{11}{20}$	$=8\frac{4}{5}$
29·00	14·5	$\frac{1}{2}$	$=8$
29·1	14·55	$\frac{9}{20}$	$=7\frac{1}{5}$
29·2	14·6	$\frac{2}{5}$	$=6\frac{2}{5}$
29·3	14·65	$\frac{7}{20}$	$=5\frac{3}{5}$
29·4	14·7	$\frac{3}{10}$	$=4\frac{4}{5}$
29·5	14·75	$\frac{1}{4}$	$=4$
29·6	14·8	$\frac{1}{5}$	$=3\frac{1}{5}$
29·7	14·85	$\frac{3}{20}$	$=2\frac{2}{5}$
29·8	14·9	$\frac{1}{10}$	$=1\frac{3}{5}$
29·9	14·95	$\frac{1}{20}$	$=\frac{4}{5}$
30·00	15	0[1]	$=0$

---

[1] This is on the supposition that ordinary atmospheric pressure is = 30 inches of mercury, or 15 lbs. per square inch.

1 inch of mercury = 13·56 inches of watergauge.

1·356 × no. of tenths ‘exhaust’ = watergauge reading.

Pressure of Air, as shown by the Watergauge.

Height in inches	Pressure in lb. to the sq. ft.	Height in inches	Pressure in lb. to the sq. ft.
0·01	0·05	0·46	2·39
0·02	0·10	0·47	2·44
0·03	0·15	0·48	2·49
0·04	0·20	0·49	2·54
0·05	0·26	0·50	2·60
0·06	0·31	0·51	2·65
0·07	0·36	0·52	2·70
0·08	0·41	0·53	2·75
0·09	0·46	0·54	2·80
0·10	0·52	0·55	2·86
0·11	0·57	0·56	2·91
0·12	0·62	0·57	2·96
0·13	0·67	0·58	3·01
0·14	0·72	0·59	3·06
0·15	0·78	0·60	3·12
0·16	0·83	0·61	3·17
0·17	0·88	0·62	3·22
0·18	0·93	0·63	3·27
0·19	0·98	0·64	3·32
0·20	1·04	0·65	3·38
0·21	1·09	0·66	3·43
0·22	1·14	0·67	3·48
0·23	1·19	0·68	3·53
0·24	1·24	0·69	3·58
0·25	1·30	0·70	3·64
0·26	1·35	0·71	3·69
0·27	1·40	0·72	3·74
0·28	1·45	0·73	3·79
0·29	1·50	0·74	3·84
0·30	1·56	0·75	3·90
0·31	1·61	0·76	3·95
0·32	1·66	0·77	4·00
0·33	1·71	0·78	4·05
0·34	1·76	0·79	4·10
0·35	1·82	0·80	4·16
0·36	1·87	0·81	4·21
0·37	1·92	0·82	4·26
0·38	1·97	0·83	4·31
0·39	2·02	0·84	4·36
0·40	2·08	0·85	4·42
0·41	2·13	0·86	4·47
0·42	2·18	0·87	4·52
0·43	2·23	0·88	4·57
0·44	2·28	0·89	4·62
0·45	2·34	0·90	4·68

Thus the Government inspector, on paying a sudden visit to the colliery, can see at a glance how things are going on. On pages 61 and 62 we give some tables of atmospheric pressures and watergauge pressures reduced to lbs. per sq. inch. In the case of the barometer tables, the supposition is that 15 lbs. per sq. inch = 30 inches of mercury, and that this barometer reading represents the normal daily atmospheric pressure. The patent barometers constructed by Messrs. Apps, of London, for me are made to work in accordance with this table. Friction in these instruments and the error due to it need not be taken into account.

Anemometer.

This instrument is used to measure the velocity of the air current in feet per second or miles per hour.

The forms of anemometer suitable for a mine are Biram's and Bagot's.

Biram's instrument records its results on a dial on the instrument like the hands of a watch. It is a very useful instrument for the firemen and overmen to carry. Bagot's automatic apparatus is for permanent use. The cups are placed at the extreme end of the mine, and the results in feet per second are printed on a strip of paper by the recorder in the fan-engine-room. Electricity is used to convey the varying speeds from the cups to the recording apparatus. Thus the manager can tell at any time or any day the speed of the air in the main return air-way, and instant telegraphic notice is given to the fan-engine-driver if the air current becomes interrupted, whilst he can at all times read the velocity of the air in the mine. The advantages of this automatic system are obvious. It is the only method of ascertaining constantly the true velocity of the air in the mine in such a way that the record is continuous, instantaneous, and cannot be tampered with. In the cups used for this apparatus the centre of each hemispherical cup moves with one-third the velocity of the air current; therefore, with four cups the speed of the vertical shaft of the measuring apparatus will represent the true velocity of the air.

The following are the rules which govern the action of a volume of air acting on a plane surface.

Let P = pressure of wind in lbs. per sq. foot.

Let V = velocity in miles per hour.

$$P = \frac{V^2}{200}$$

$$V = \sqrt{200\,P}$$

When the plane surface is inclined at an angle e to the direction of the wind, then let the *horizontal pressure* be represented by Ph.

Then $Ph = P \sin 1{\cdot}842 \cos e$.

Let the *vertical pressure,* i.e. the pressure at 90° to the wind, be represented by PV.

Then $PV = P \operatorname{cotan} e \sin e\, 1{\cdot}842 \cos e$.

Pressures and Velocities.

Miles per hour	lbs. per square foot	Miles per hour	lbs. per square foot	Miles per hour	lbs. per square foot
1	·005	35	6·125	69	23·805
2	·020	36	6·480	70	24·500
3	·045	37	6·845	71	25·205
4	·080	38	7·220	72	25·920
5	·125	39	7·605	73	26·645
6	·160	40	8·000	74	27·380
7	·245	41	8·405	75	28·125
8	·320	42	8·820	76	28·880
9	·405	43	9·245	77	29·645
10	·500	44	9·680	78	30·420
11	·605	45	10·125	79	31·205
12	·720	46	10·580	80	32·000
13	·845	47	11·045	81	32·805
14	·980	48	11·520	82	33·620
15	1·125	49	12·005	83	34·450
16	1·280	50	12·500	84	35·280
17	1·445	51	13·005	85	36·125
18	1·620	52	13·520	86	36·980
19	1·805	53	14·045	87	37·845
20	2·000	54	14·580	88	38·720
21	2·205	55	15·125	89	39·605
22	2·420	56	15·680	90	40·500
23	2·645	57	16·245	91	41·405
24	2·880	58	16·820	92	42·320
25	3·125	59	17·405	93	43·245
26	3·380	60	18·000	94	44·180
27	3·645	61	18·605	95	45·125
28	3·920	62	19·220	96	46·080
29	4·205	63	19·845	97	47·045
30	4·500	64	20·480	98	48·020
31	4·805	65	21·125	99	49·005
32	5·140	66	21·780	100	50·000
33	5·445	67	22·450		
34	5·780	68	23·120		

Let the *normal pressure* = *Pr*, that is to say, *Pr* represents the pressure at 90° to the surface opposed to the wind.

$$Pr = \frac{Ph}{\sin e}$$

A pressure of 1 lb. per sq. foot = 4,881 grammes per sq. metre, so that to reduce lbs. per sq. foot to grammes per sq. metre, multiply by 4,881. A table of pressure and velocities is given on the preceding page.

The Barometer.

The Coal Mines Regulation Act requires this instrument to be kept at the top of every pit where dangerous gas has been found. We advise an aneroid barometer to be added underground, since miners can read this form of the instrument more easily, and the firing of shots will not interfere with the internal mechanism of the barometer. If a mercurial one is used, when purchasing it, it is best to obtain its correction for capillarity from the maker. The difference of level being frequently got by means of an aneroid, owing to the difference of readings at the two points of observation, it may not be out of place here to give the formula for arriving at the required result. The table to find the value of K may be found in Molesworth's 'Pocket-book of Engineering Formulæ,' page 13. The values given there are to be *added*.

Let R = reading in inches on barometer at lower station.

r = reading at upper station.
T = temperature of lower station.
t = ditto of upper ditto.
K = correction necessary due to $T + t$.
(For this value, see table referred to.)
H = difference of level in feet between the two stations.

Temperature	Correction	Temperature	Correction
° 32	0·00	° 62	0·08
33	0·01	63	0·09
34	0·01	64	0·09
35	0·02	65	0·09
36	0·02	66	0·09
37	0·02	67	0·10
38	0·02	68	0·10
39	0·03	69	0·10
40	0·03	70	0·10
41	0·03	71	0·11
42	0·03	72	0·11
43	0·04	73	0·11
44	0·04	74	0·11
45	0·04	75	0·12
46	0·04	76	0·12
47	0·05	77	0·12
48	0·05	78	0·12
49	0·05	79	0·13
50	0·05	80	0·13
51	0·06	81	0·13
52	0·06	82	0·13
53	0·06	83	0·14
54	0·06	84	0·14
55	0·07	85	0·14
56	0·07	86	0·14
57	0·07	87	0·15
58	0·07	88	0·15
59	0·08	89	0·15
60	0·08	90	0·15
61	0·08		

Then $H = 60{,}000\ (\log R - \log r)\ K$.

This is approximate—

To correct the reading of the barometer for temperature use the table on preceding page.

In a mine the value of K may be left out.

The barometer must always be closely watched, and any difference between the readings at the bottom of the upcast, where a barometer should be stationed, and the reading at the foot of the downcast, above that naturally due to difference of level between the two shafts, should be the signal for a searching examination for falls of roof or misplaced air-doors being made without unnecessary delay.

Thermometer.

It has been already shown that the head of air, or motive force of the ventilating column, is due to its temperature when a furnace is in use. So many measurements are given in French or Belgian works on mining that I have introduced formulæ for interchanging the temperature readings in the different scales, to assist the reader.

Let F = No. of degrees Fahrenheit's scale.
„ C = No. of „ Centigrade „
„ R = No. of „ Réaumur's „

$$F - \frac{9^c}{5} + 32$$

$$F = \frac{9\,R}{4} + 32$$

$$= C + R + 32$$

$$C = \frac{5(F—32)}{9}$$

$$R = \frac{4\,(F-32)}{9}$$

Freezing point = 32° F.
,, ,, = 0° Centigrade and Réaumur.
Boiling point = 212° F.
,, ,, = 100° Cent.
,, ,, = 80° Réaumur.

In pits containing brassy coal, great care must be taken to keep the temperature of the air-ways as low as possible to avoid spontaneous combustion. Again, it will be seen that a sudden rise of temperature will cause the coal to give off a certain volume of gas due to its expansion. Gases expand nearly uniformly with an increase of their temperature, and when the volume of permanent gases remains constant, the pressure increases nearly uniformly with the increase of the temperature. For instance, according to Regnault, let us suppose a volume of air to be represented by 1 at a temperature of 32° F., the volume is increased to 1·36706 at 212 F.; that of hydrogen to 1·36613, and that of carbonic acid to 1·37099. The laws that govern this important feature of gases, viz., their expansion under an increase of temperature, may be deduced from the following.

Absolute Temperature.

It has been assumed that since air expands as its temperature increases, so would it on the other hand diminish its volume if the temperature fell; thus if at 32° F. air has a volume of 1, it is assumed that on arriving at a temperature of 493°·2 below freezing point, 461°·2 F. below zero, the air would be so attenuated as to be in a state of collapse, and would possess no elasticity. This point is called *absolute zero*, and the temperature of any body measured from absolute zero is called the *absolute temperature* of that body.

Thus if a given quantity of air at 0° F. be raised to + 461° F., it is (if under a constant pressure) expanded to twice its original volume; if heated to 922° F. its volume will be trebled.

Therefore the laws to be borne in mind are—

i. The pressure of air or any gas varies inversely as the volume when the temperature is constant.

ii. The pressure varies directly as the absolute temperature when the volume is constant.

iii. The volume varies as the absolute temperature when the volume is constant.

iv. The product of the pressure and volume is proportional to the absolute temperature.

The *absolute zeros* of the three thermometer scales are—

Fahrenheit	461°·2
Réaumur	219°·2
Centigrade	274°·00

But, to avoid excessive labour in calculation, it will suffice to drop the decimal reading. Gas compounded of carbon and hydrogen occasionally throws out oily products, which will interfere with the action of these laws.

It will be seen, therefore, that violent changes of temperature in ventilation are to be avoided in working fiery and brassy mines, or gas will be liberated that otherwise would have remained in the seam.

The specific gravity of the gases found in coal-mines is as follows:—

		Weight of 1 cubic foot.
Air is represented as	= 1·00	
Hydrogen ,,	= 0·069	·0056
Carburetted Hydrogen	= 0·550	
Steam	= 0·620	
Nitrogen	= 0·976	·078
Olefiant or marsh gas	= 0·980	·079
Air	= 1·00	·08072
Oxygen	= 1·100	·089
Sulphuretted Hydrogen	= 1·190	
Carbonic acid (choke-damp)	= 1·520	·123
Sulphurous acid	= 2·120	

The weight per cubic foot is given here in decimals of 1 lb. avoirdupois.

CHAPTER VII.

GENERAL OBSERVATIONS WITH REGARD TO VENTILATION IN PRACTICE.

VENTILATION in theory will be found a comparatively easy matter; in practice, however, falls of roof and the carelessness of the miners will be found a source of endless annoyance and anxiety. A miner is bound by law (Sec. 55 Coal Mines Regulation Act) to *report instantly to the fireman* should he discover anything wrong with the ventilating current, or the *state* of the roof or *mine generally*, such as leads him to suspect danger to himself or his fellow-workmen. This is not always done, as the Act requires, *instantly*, and the duty of every manager is plain where this is the case; the Act having been broken, the miner should be punished under the Act. Such very strong assertions are made concerning the management of collieries by men, whose very assertions show their ignorance of the subject, and since such assertions, being those of uneducated men, are always directed more or less against the capitalist or employer of labour because he is an employer of labour, that we strongly recommend managers to adopt two practices, viz., to summon *all offenders* before the magistrates, and to register the conviction in a book kept for the purpose; for it will be seen at a glance that in the present day none but very experienced viewers are acting wisely if they take upon themselves the office of weighing the culpability of the offender. A reference to the various Mining Acts will show that wilfully damaging ventilating machinery is expressly provided for. It is not the wilful damage, but the damage and danger incurred *by carelessness*, that the manager will have to provide against.

The position of all important air doors in a mine may be electrically determined by a piece of apparatus designed by the writer for the purpose, which costs but a trifling amount; but the 'trapper' system, by placing boys to watch the doors, is perhaps better, only it will frequently be found necessary to place men to watch the boys.

The most important posts in the ventilation department of a colliery are the fireman, roadsman, trapper or doorkeeper, and master wasteman.

The fireman's duties are very hazardous. He is a *competent person solely employed to test the pit for gas.* When inflammable gas has been found (and we presume that all viewers will see the propriety of examining before each shift begins work, even where it has not been found), he has to examine the pit once in every shift or once in every twenty-four hours; should he find gas, he must report the same in a book kept for the purpose. The Act should have made him post a notice at the pit-head, containing extracts from the book, showing briefly where gas had been found throughout the mine.

He also places 'fire boards,' or notices of dangerous gas, at the entrance to headings which have been found in his examination to contain it. These boards should be painted red, and made easily recognisable to miners who cannot read. Another most responsible duty of the fireman is to act as the 'competent person' where shots are being fired. No shots should be fired where naked lights are used in the vicinity, as a large volume of gas may exude or be discharged after the shot, and so become ignited, although the ventilation may be ample; neither should lamps on Davy's principle be used for the operation, but self-extinguishing lamps, such as Stephenson's or Williamson's safety lamps.

He should commence firing his shots on the return side of the pit, always facing the air current; thus the gas liberated by the first shot is being blown away by the air current from the site of the next one he is going to fire. This is a most important point, and should be strictly adhered to. He should also examine the sides of props near the working faces, to see if they bear the marks of matches being struck on them by the colliers.

The fireman should make his inspection immediately before the entrance of the workmen to the mine, *and allow no delay to occur*, as a large volume of gas may be easily discharged in the interval elapsing between his inspection and their commencing work, which may result in an accident.

He should take notice of any loose hay or straw lying about in the pit, and report on any horse-dung or rubbish left lying about on the roadways; if brassy lumps of coal are thrown out on the road, they should be got rid of. These are all apt to generate gob-fire by spontaneous combustion; and although such combustion may not be sufficiently active to cause flame, nevertheless it must be remembered that all combustion detracts from the purity of the air, by depriving it, as already shown, of a portion of its oxygen, and by increasing the temperature of the air, it tends to facilitate the escape of noxious gases.

The roadsman's duties principally consist in the maintenance of all the air-ways in a good and secure condition. His especial duty is to provide against falls; and should these falls occur he must exercise *great care*, both in approaching the spot after the fall, and in providing against a further fall. On no account must he ever work with a naked light, and it will be found advisable to give him a practical training as a fireman before he becomes a roadsman. This is easily done by sending him round the pit with the fireman for a time, so that he may become conversant with the nature and behaviour of the fire-damp, and the necessary precautions to be taken in testing for its presence with the lamp. Possibly at a future date, when the mania for education has been diluted with a little common sense, the school boards in mining districts may see the propriety of teaching the pit boys the nature and behaviour of fire-damp, &c., but in the meantime the manager must do it.

The overman is responsible (subject to the control and supervision of the manager) that adequate ventilation is supplied to all parts of the pit; and these three officials, viz., the overman, fireman, and roadsman, should be selected from the others on account of their qualifications to fill these respective posts, and not on account of their long service or good conduct: the posts are too responsible for this latter cause of selection, which is only applicable where no responsibility exists, and not where the person to be promoted is incapable of accepting that responsibility.

Falls of roof can only be provided against by timbering of suitable strength and quality; this cannot well be entered into here, as the nature of the roofs of no two pits is alike. It will be found advantageous to study the methods of timbering and walling in use in the district under the various conditions of working, and to apply that system which appears to meet your own case best.

A practice used to exist of so splitting the main air current, that in the first case the greater part of the air circulated round the pit, visiting district after district, thereby becoming more impure and explosive as it went on; and in the second case, the splits were so badly arranged that some districts were never really ventilated at all. Care should be taken that this does not occur.

The master wasteman is a most responsible person, and the waste, or old workings, is a very fruitful source of gob-fire, the cause of which we will here explain, and show how it may be provided against.

'Brassy coal,' as it is technically called, is coal containing a certain per-centage of iron pyrites. This pyrites is the yellow spots we have on house coal. Its chemical formula is FeS_2. It is therefore sulphide of iron, and merely requires one more atom of oxygen to allow the sulphide to become converted into a sulphate with the evolution of a considerable quantity of heat. If coal becomes exposed to a moist hot atmosphere, or if air is blown *through* the coal instead of over the *surface* of the coal, the cooling effect is partially destroyed, and the heat evolved by the oxidation of the pyrites is sufficient to fire the seam. In waste workings this frequently takes place, and the following rules must be observed. No slack coal or waste heaps must be allowed to accumulate, neither must the practice of 'putting back' the small coal be permitted. The air circulating in the waste workings must be a brisk dry current, or the waste had better be 'stopped off' altogether, but in no places where pillars have been removed must the air be allowed to stagnate; if these precautions are taken gob-fire will be guarded against as far as human foresight can guard against such contingencies.

This is the master wasteman's province, and he should have frequent interviews with the manager, and discuss the state of the ventilation from day to day in such wastes; but as a fixed rule it is advisable always to *stop off all goafs and wastes by air-tight fireproof brick stoppings*, as described in a previous chapter, and pack them up.

In mines, the inclinations of the seams of which are very steep, it is obvious that, if possible, the intakes should be on the dip side of the mine, and the return air-ways on the rise side, in order to utilise the natural ventilative effect that would be thus brought about by the heated air and diluted gas being lighter than the air in the intakes, thus causing a natural draught through the workings. We strongly urge the filling up of all waste workings as much as possible. They should be looked upon as so many gasometers, and dispensed with whenever possible. It is in these large wastes that the danger of a low barometer reading exists. In mining a seam with another a short distance below it care must be taken to prevent the gas from the lower seam blowing into the upper one. Explosions have occurred from this, and the distance and nature of the strata between the two should be proved with a borehole if the existence of a lower seam is known or suspected; *the hole must be closed when the object is attained.*

Another system of ventilation, by means of jets of steam or air, has been tried, but is a very wasteful system except under peculiar circumstances. Where a hauling engine is situated directly at the bottom of the shaft, the exhaust steam may be utilised, and it is only under such circumstances as these that it can be employed as a blast to cause a circulation of air.

It has been calculated that the *vis viva* of a jet of air under a compression of five atmospheres only represents $\frac{6392}{12851}$ of the force necessary to compress it. This loss may be diminished by lowering the degree of compression in tension.

The following are the elements of success in using blasts as a means of ventilation, and are the results of experiences in the experiments of Nozo and Geoffroy. The blast pipe should be seven times longer than its diameter; the distance of the mouth of the tuyère, from the point of entrance to the blast pipe, should not exceed three times the diameter of the blast pipe, and the tuyère should not penetrate to a greater extent than twice the diameter of the blast pipe. It is immaterial to the effect produced whether the position of the blast pipe be vertical or horizontal. Practical experience shows that the useful effect obtained by ventilating with a steam blast, is only from 5 to 6 per cent. of the useful effect produced by a machine absorbing the same amount of steam.

To utilise a jet of air to ventilate a mine, the best effect will be secured by placing the delivery or blow pipe in the centre of the axis of the shaft. It is obvious that the inertia that the compressed air possesses is spent in opposing the resistance of the column of air in the shaft, and in to a small degree communicating a portion of its inertia to that column, consequently the *velocity* of the air in this case is of considerable importance. In fact, the power required to compress the air to effect the blow at the necessary velocity is greater than the force required to impart that velocity by means of a Guibal fan, and this should be sufficient reason against the use of compressed air or steam blast as a means of ventilation for a mine.

When steam at high pressure impinges against a column of air of a lower temperature, a considerable portion of the useful effect of that steam must be lost by reason of the unavoidable condensation that takes place; and the increase of temperature that this condensed steam is able to impart to the air column in the shaft, had it remained at its original pressure, is lost by its condensation, so that the atmosphere of the shaft in which the blast is fixed and delivered is in one place nothing more than a mixture of air and steam of a nearly uniform temperature, and it is for the engineer to determine the precise spot in the shaft at which this state of things is brought about; if at any depth in the shaft good ventilation cannot take-place.

The general conclusions arrived at are as follows:—

Blasts of air, as a means of ventilation in collieries, may be dismissed from our consideration as impracticable.

Furnaces and steam jets must not be used, except when such advantageous circumstances as we have pointed out permit their economical use.

Mechanical ventilation by means of fans, which do not shut off the communication between the surface atmosphere and that of the mine when the fan is not in motion, whether a direct-acting blowing fan or a centrifugal fan, is not a satisfactory system of ventilation, on account of the great speed at which the fan must be rotated. Of this class, Guibal's fan, when properly erected, is the least objectionable, and may fairly be regarded as an economical and convenient method of ventilating a pit; Lemielle's air-pump system is an expensive and easily deranged system, and on the whole we can safely recommend Guibal's fan and Schiele's fan as being the best means of mechanical ventilation, although from an engineering point of view they cannot be considered perfect.

Practical Notes on Ventilation in Long Wall and Rib and Pillar Working.

Long Wall.

The practical advantages derived from long wall working as against rib and pillar working may be thus briefly summed up:—

14 per cent. more large coal is obtained by the former than the latter.

19 per cent. average more coal per acre is obtained.

It is applicable when the seam is not faulty, when it is thin or containing an ironstone seam workable with the coal, as in North Staffordshire, or a seam with a dirt parting. Less brattice cloth is required for ventilation. It permits of shot firing with greater safety.

Rib and Pillar.

A thick seam, like the ten-yard coal in South Staffordshire, a pit that is not fiery, and where the coal seams do not lie at a considerable inclination.

Local circumstances tend to influence the adoption of this system of working.

It is confidently asserted that explosion is less frequent with the long wall than with the rib and pillar system of working; and when we consider that most of the thick coal seams, where rib and pillar work is in force, are not fiery, we incline strongly to the opinion in favour of long wall when practicable. The large goafs that occur in pillar work are thus avoided, and the distribution of air is more uniform. It is easier to head out to the colliery boundary, and 'work home' to the pit with long wall work, and the disruption of the surface is less.

In shot firing the fireman should fire from the return side of the air current towards the intake, so that he may always have the gas he may liberate with a shot blowing from the position of the next one he intends to fire. In opening out a pit for rib and pillar working, the first step is to connect the two shafts by headings to secure ventilation. Then the levels or way gates are set off at right angles to the direction of the dip of the coal seam to form a water level.

Two roadways are driven from the hauling shaft, one in each direction; these are the main gate roads through which the coal won is brought to the pit bottom.

If it is a dry seam the water level need not be driven on the dip side.

It is very necessary that the main air roads at the bottom of both up and downcast shafts should be brick-arched when they have settled.

When driving the headings great care must be exercised in ventilating them by cut-throughs or passages connecting the intake heading with the return heading, and capable of being stopped up afterwards with brickwork; if this cannot be done the heading must be divided with brattice cloth, the air current going up one side and down the other.

In working the ten-yard coal by rib and pillar the sides of the work require to be divided from each other by 'fire ribs' in partitions of coal; a road is driven up each of these ribs between two stalls, and a hole cut through into the stalls when required. This secures the escape of any gas. An examination of the two systems in practical work will afford the reader more knowledge and insight into the question than a description here.

Waddle's fan and Schiele's fan are fans of recent date. For a small colliery Schiele's fan is effective. A practical hint may be given to those who design fans. The temperature of the air to be withdrawn from the mine must be considered. Many fans of Guibal's pattern do not act efficiently owing to this point being neglected. It is obvious that the volume of the ascending air being increased by its temperature, the exhaustive capacity, or power the fan has of producing a given depression, must be allowed to be sufficiently great to admit of this increased volume being held by the fan.

Sir W. Fairbairn has stated that it was practically impossible to circulate 250,000 cubic feet of air per minute with a vacuum fan. Allowing this to be so, the result of experience shows that where an adequate result can be obtained the fan should be used in preference to the furnace on the score of safety. The question of the ultimate limit of the power of a fan to circulate air does not in our opinion rest so much on the fan, as upon the solution of the question of the number of shafts. We are of opinion that coal is more safely got, and in some cases at less hauling expense, by having more shafts, than with a single upcast and downcast shaft only.

If compressed air becomes the power for underground haulage, we must design our fans for giving a high velocity to a large quantity of air, and depend on the exhaust air from the engines underground for the volume.

It is impossible to lay down a hard and fast rule on this point. The tendency is to supplant steam and horses underground with compressed air as a motive power. The result will be that questions will arise and new difficulties present themselves that it is nearly impossible to foresee; all the mining engineer has to provide against is not to follow the steps of his predecessors, and so fall into their errors when they were in error, until he has satisfied himself of the probable results, by means of the experience and scientific knowledge brought to bear upon the subject in the present day. Cheapness of production is the essential qualification of a good colliery, and the change from horses and steam to compressed air may to a certain extent act as a natural check upon the effect of short hours and advanced wages. This point should not be overlooked in laying out a new colliery. We have therefore given the following as points to be considered when using compressed air as a means of conveying power to a distance.

Compressed Air.

That underground haulage by means of horses will be eventually superseded by atmospheric pressure, either used directly as an air locomotive, or indirectly as a hauling engine, actuated by compressed air in place of steam, very few persons will doubt. It affects the subject under our consideration indirectly. Steam underground in fiery mines is a source of danger; horses are as expensive when at rest as when working—they consume, as we have already seen, 6·3 times the amount of air required for a man per minute. On the other hand, in compressed air we have a power that costs nothing when at rest, that is perfectly safe in the most fiery mine, and that largely increases the *quantity* of air in the mine without increasing its *final velocity*, whilst to a certain extent lowering its temperature. These considerations cannot fail to attract the attention of modern mining engineers. We propose to give here the main features of interest or importance that require to be taken into account with this method of transmitting power.

Firstly, from experiments at the Powell Duffryn collieries we find that a low degree of tension gives the best effect; that is, that it is not advisable to compress the air to too great a pressure, as the following results show:—

Air pressure	40	lbs. per sq. in.	yielded in useful effect	25·8	p.c.
,,	34	,,	,,	27·1	,,
,,	28·5	,,	,,	28·5	,,
,,	24	,,	,,	34·9	,,
,,	10	,,	,,	45·8	,, .

M. Colladon, one of the engineers of the St. Gothard Tunnel, made a minute of experiments on compressing pumps (‘Annales industrielles,’ 1877, vol. ii., col. 51), the pressures varying up to ten atmospheres (150 lbs. per square inch), with the piston speed exceeding 201 feet per minute, and the quantity of water injected being $\frac{1}{1000}$ the volume of the stroke, the temperature of the air 60° to 80° F. Here the stroke was 17·3 inches, the volume of clearance being $\frac{1}{80}$ that of the stroke.

The injection of water cooled the air economically and effectively.

The useful effect of air thus compressed and converted into mechanical power is at first sight low in comparison to the absorption of steam power to compress it, indeed, as low as only 17 per cent., and as high as nearly 60 per cent. The causes of these variable results are found when we examine the following points.

Variable piston speed of the compressing engine produces fluctuations and waves of compression in the delivery pipes. These waves of air at variable tensions act and react on each other. Again, the power absorbed in compressing reappears in the receiver or air tank and air cylinders of the compressing pump, in the form of heat; but we have seen that if air at a temperature of 0° F. be heated to 461° F. its volume is doubled; thus if we were to get a temperature of 461° F. in the air cylinders, we should have to compress just double the quantity of air per minute that we should if we could keep the temperature down by artificial means to 0° F. Air, when compressed to four times its original pressure, would gain 258° F. in temperature; the volume would increase about 2 : 3; the absolute power absorbed by the production of this 258° of heat would be about 34 per cent. of the total steam power; where water was injected as a spray, at a sufficiently low temperature and in a volume sufficient to reduce the air temperature in the air cylinders during $\frac{7}{8}$ of the stroke, this loss of 34 per cent. could be saved, and therefore still available as power. The next point is that the higher the temperature and pressure in the air receiver, the lower will it be at the air-port or orifice of the air engine, and the snow ice will be found causing stoppage by blocking up the exhaust. The remedy is to cool the air to about 85° F., and lubricate the working parts of the air engine with glycerine. Proper and sufficiently large ports will do much to prevent the formation of icicles. Of the 480-horse power developed in the steam cylinder of the air-compressing engine at the

Powell Duffryn collieries to drive twenty-six compressed air engines, about 25 per cent. was represented by the work of these engines. We maintain that there is too much stress laid upon this loss of power; the advantages of the use of compressed air in mines from the economical and safety point of view is forgotten in the endeavour to compress air at a point A with a known power of, for instance, 25 H.P., and reproduce it as a power of 25 H.P. at B. Eminent engineers have fixed the limit of effect at 50 per cent. of the steam power, but we venture to predict that at 45 lbs. per square inch air pressure, 85° F. temperature, and large hauling ropes, with a proper attention to the state of the roads, curves, and gradients, that limit will ultimately be exceeded. Mr. Bramwell stated in 1874 that taking air at 40 lbs. per square inch and water at 700 lbs., the standard limit of pressure advocated by Sir William Armstrong, and applying these two means of power to an engine, it would require 17½ times as great a volume of air as that of water to produce the same results with the engine with the two different agencies, and the velocity of flow would have to be 17½ times as great with air as with water; as there was 17½ times a greater bulk of air than of water to deliver through the conducting pipe, the power required to overcome the friction would be as the cube of the velocity, 5,359 times the power required with water, supposing the density of the air and the water to be the same; but air at 40 lbs. pressure is about 220 times as light as water, so that the result should be divided by 220, giving 1·4 times greater friction and 24 times greater loss of power with air than with water. If the conducting pipes are large, and the velocity of the air slow, the relative loss of power from friction will be very much less. Water is no use in ventilation in a mine where air is of very great service, so that what applies on the surface must not be used as an argument against the use of compressed air underground.

Mr. Daniell's experiments published in the *Transactions* of the Institution of Mechanical Engineers in 1874, when tabulated from the 'Indicator Diagrams' give the following. The air was compressed with 70 lbs. steam pressure cut off at $\frac{1}{14}$ the stroke in a non-condensing engine. The theoretical mean effective steam pressure is 35·8 lbs. per square inch. One cubic foot of steam at 70 lbs. pressure being taken at 0·183 lb., the theoretical useful effect per lb. of water evaporated $= \frac{35{\cdot}8 \times 144 \times 4}{0{\cdot}183} = 112{\cdot}682$ foot pounds in the steam cylinders of the compressing engine. The theoretical useful effect of air at 40 lbs. pressure is 29·072 foot pounds per pound of water evaporated, the per-centage of useful effect actually yielded by the air being 25·8.

The greatest attention should be paid to the design and proper construction of the air valves of

Useful effect of compressed air compared with steam at five different pressures. (Daniell's experiments.)

Pressure of air in receiver in lbs. per sq. inch above atmospheric pressure	40·0	34·0	28·5	24·0	19·0
Weight of 1 cubic foot of steam in lbs.	0·123	0·110	0·099	0·090	0·079
Theoretical useful effect of air in ft. lbs. (F.)	29072	30527	32114	39326	51608
Theoretical useful effect of steam at the same initial pressure air in ft. lbs. (G.)	59544	56455	52424	48393	43392
Percentage of useful effect from air compared with steam $(100\times\frac{F}{G})$	48·8	54·1	61·3	81·3	118·9

an air engine to secure free inlet and egress. The water injected into the expanding cylinder should be of such a temperature that the heat taken from the air whilst compressing it, should be applied again to the air when in the expanding cylinder as nearly as mechanical inventive genius can arrange. Pressure gauges should be attached at convenient spots to show the pressures existing along the line of pipes which convey the compressed air from the compressor to the air engine underground if it be a stationary engine. Conducting pipes, from 2½ to 3½ inches inside diameter, will be found sufficient, except under special circumstances. A high velocity of the air will be produced if the pipes be too small; the point to be aimed at being that no more air shall pass through the conducting pipes than is required by the engine per minute. The discharge of the exhausted air should of course be in the direction of the air current, when possible into the main return air-way, in preference to an intake, with a view to cooling and still further diluting the return air current with fresh air, without detracting from the amount travelling up to the working forces by the intakes.

The question of the loss of head due to friction has been carefully considered at the Ronchamp collieries. From experiments conducted there it has been deduced that the resistance is directly as the length of the conducting pipe; it is directly as the square of the velocity of the flow, and inversely as the diameter of the pipe. The point to be noticed is that for velocities *under four feet per second* the loss by transmission is not very great, but with higher velocities it rapidly becomes an insurmountable object. The following formula will be of use in the above question:—

Let A be the area of the pipe in square feet.

S = the perimeter multiplied by the length of the pipe in feet

V = the velocity of the air in feet per second.

Taking the value of f at $\frac{1}{4}$ of the friction in the air-ways of a mine.

h = the head or height of the motive column in feet, the pressure due to which must be subtracted from that in the reservoir to find the pressure at the end of the air-pipe.

Then—

$$h = \frac{F S V^2}{a}$$

Practically, large pipes for conducting the air down the shaft, tight joints, and an even temperature are the points to be noticed in conducting air from the surface to the hauling engine or air tank underground.

In cases of water-pressure engines used to work mechanical haulage, their power must be thus calculated :—

Q = quantity of water in cubic feet per minute.
H = head of water in feet.
P = effective horse-power.

$$Q = \frac{661\ \mathrm{P}}{H}$$

$$\mathrm{P} = \cdot 001512\ H$$

It is essential that the velocity of the water in the supply pipes should not exceed 400 feet per minute.

CHAPTER VIII.

MINES REGULATION ACT.

SUBJOINED is an abstract of the duties, as set forth by Act of Parliament, of those persons who are responsible for the ventilation of the mine.

Reports, where danger exists, cannot be made too promptly, and it is the duty of every man in the pit to cause no danger to himself or others, and when danger exists, such as a fall of roof or a blower of gas, he should not delay to report the same at once, but should place a warning at the spot, or station some one to caution persons from inadvertently entering the spot where the blower or fall is.

1. An adequate amount of ventilation shall be constantly produced in every mine to dilute and render harmless noxious gases, to such an extent that the working places of the shafts, levels, stables, and workings of such mine, and the travelling roads to and from such working places, shall be in a fit state for working and passing therein.

2. During the twelve months following the discovery of inflammable gas in a mine, a competent person shall, before the time for commencing work in any part, inspect with a safety lamp that part of the mine and the roadways leading thereto. The inspection shall be made once in every twenty-four hours if one shift is employed, and once in every twelve hours if two shifts are employed; and a

report of the state of the ventilation shall be recorded in a book kept at the mine for that purpose, and signed by the person making the same. In a mine in which inflammable gas has not been discovered during the preceding twelve months, the inspection shall be made once in every twenty-four hours.

3. Whenever the mine, or any part thereof, is discovered to be dangerous by reason of gases prevailing therein, or of any other cause, every workman shall be withdrawn from the mine or part thereof, and they shall not be readmitted until the mine or part thereof has been inspected by a competent person, and reported by him to be safe.

4. In every working approaching any place where there is likely to be an accumulation of explosive gas, no lamp or light other than a locked safety lamp shall be used, and whenever safety lamps are required, a competent person shall examine every safety lamp immediately before it is taken into the workings for use, and ascertain if it be secure and securely locked; and in the said part of a mine a person shall not, unless he is appointed for the purpose, have in his possession any key or contrivance for opening the lock of any such safety lamp, or any lucifer match or apparatus of any kind for striking a light.

Overman.

Subject to the control and supervision of the manager, the whole operative details shall be under the care and charge of the overman. The overman shall see that the workmen of every class, in their several departments, discharge their duties; and shall receive and attend to all reports made to him as to the state of repair of the air courses, machinery, mid-wall trap doors, roads, cubes, and working places. He shall cause remedies to be provided where needed, and shall see the general and special rules faithfully and vigorously enforced, and he shall have power to hire and discharge workmen.

He shall have under his immediate and special charge, the shaft, slides, pump, and relative fittings, all of which he shall keep safe and efficient.

He shall attend to the ventilation, in terms of rule No. 1; and to the observance of other general rules, as far as these, from their nature, can be observed by himself, or fail to be observed by others under his charge.

He shall perform the special duties as to the examination of machinery and others.

He shall see that a plentiful supply of timber for props and other purposes, required by workmen to carry on their operations with safety to themselves, is always ready; and shall cause the same to be cut in proper lengths, and laid down in the working places.

He shall, without delay, report to the manager, or agent, or owner, any matter or thing coming under his notice, necessary to be observed or carried out, with a view to compliance with the general or special rules, which he cannot perform.

Furnacemen.

Subject to the control of the manager or overman, the furnaceman shall act under, and obey the directions of the engineman.

Roadsmen.

The roadsmen in their different divisions and shifts, if more than one, shall, at least daily, make careful inspection of the whole roadways and working places, from the pit bottom throughout the mine, and shall keep the same free from all obstructions, and of the fixed height and width necessary for proper passage and for ventilation. They shall repair and remedy all damages and defects in the roads of the mine, and shall examine, and put and keep in proper condition, all trap doors, and see the regulations enforced that the same are kept closed, and whenever practicable should endeavour to make and keep such trap doors self-acting. They shall make and place sufficient trap doors whenever the progress of the operations of the mine shall render these necessary.

They shall stop the passage of men and materials through or under defective roads, roofs, or places, until the necessary repairs shall have been executed. They shall receive information concerning any interruption in the ventilation or any other cause of danger, and shall communicate with the manager or overman immediately, and shall aid and assist in the rectification and remedy of the same, and shall, when so employed, be permitted to use only safety lamps in mines where inflammable gas has been found within the preceding twelve months. All lighted or combustible substances are forbidden to be used in the course of such operations.

The roadsman shall report to the manager or overman any instances of neglect on the part of miners, in not carrying forward their faces or walls, in accordance with the plan pursued in working the mine. They shall also examine and report to the manager or overman instances of neglect, and acts of carelessness on the part of the miners or brushers, in failing to remove, or in not removing with proper caution, the strata necessary to be removed to form roads, or in not carrying forward the brushing with sufficient regularity and of the proper dimensions, or with leaving the brushing with loose or hanging stones in and about the strata 'brushed.'

As removing falls from the roofs of roads and air courses, and repairing defects therein, are within the roadsman's duties, and as they are charged with the maintenance of all roads and passages in the mine, they are enjoined to proceed with the greatest caution —both for their own safety and the successful execution of their duties. In these operations they must therefore be careful, and are required to prevent all other workmen coming near any defective places, or interfering with them when at work. They are required to undertake no repair of unusual magnitude or danger without sufficient assistance, and until provided with every necessary material, which will be supplied on application to the manager or overman.

Without prejudice to the foregoing directions, it will be the special duty of the roadsman to observe the matters embraced in the following general rules:—

1. Upon discovering that any part of the mine is dangerous, to withdraw therefrom any workmen therein employed (which workmen shall be subject to the roadsman's orders, to that effect), and report the state of matters to the manager, overman, or fireman.

2. To report to the manager or overman where they observe any violation of the rule as to the use of gunpowder.

3. To see that every man-hole, or place of refuge, is kept clear.

4. To examine the roof and sides of every travelling road at least daily, and see that the roof and sides are safe.

5. To report any damage to any fence, casing, lining, means of signalling, or otherwise, that may be observed.

Firemen.

If inflammable gas have been found in the mine within the preceding twelve months—which it shall be the duty of the manager to ascertain and intimate to the fireman—then it shall be the duty of such fireman, or firemen if more than one, in their respective positions or shifts, to perform the duties of examining and inspecting, with a safety lamp, the mine and roadways, and making the report thereon in terms of Rule No. 2.

If the fireman shall encounter falls from the roof in any of the roads which he requires to traverse, or in working places under the care of the miners, he shall not proceed further in the direction of such falls, so as to pass under the broken roof, but shall endeavour cautiously to ascertain if there be any accumulation of fire-damp or other impurity in, about, or beyond the falls, so that the safest way of clearing the same may be learned ; and he shall proceed elsewhere through the mine to examine the unobstructive parts thereof, and to complete his inspection; whereupon the fireman shall report to the manager or overman the state of the falls, and whether free from impurity, to the end the necessary directions may be given for having the same cleared away, and the roof secured ; and until this shall be done, no miner or workman shall be at liberty to proceed near, or be under the broken roof unless employed in remedying the same.

Doorkeeper (trapper).

Every person employed in keeping a trap door, for the production or promotion of ventilation in the mine, shall, during his shift, remain continually at the post assigned him, and carefully observe the directions he receives from the manager, overman, roadsman, or fireman, as to the opening and shutting of such doors, either on the occasion of other workmen passing through the same, or at the beginning or end of the shift.

Coal-dust and Safety Lamps.

By the kindness of Mr. Galloway, I make here some extracts from his paper 'On the Influence of Coal-dust in Colliery Explosions,' read before the Royal Society. On this important subject Mr. Galloway says: 'In all dry mines the coal-dust lying on the floor of the roadways rises in clouds and fills the air when it is disturbed by the passage of men, horses, small waggons, &c. A sudden puff of air, therefore, such as that produced by a local explosion of fire-damp, or by a shot blowing out its tamping, must necessarily produce the same effect in a greater or less degree according to its intensity. The mixture of coal-dust and air, formed by the action of either the fire-damp explosion or the blown-out shot, will be inflammable if it contain any larger proportion of fire-damp than 0·892 per cent.; and the flame of the original explosion will pass on through it, extending the area of the disturbance as far as the same conditions exist, or, it may be, to the utmost limits of the workings. If it contain more than 0·892 per cent. of fire-damp, it will be more and more explosive, according as the proportion of fire-damp is greater, until a maximum point is reached, beyond which its explosiveness will begin again to decline. If, lastly, it contain less than 0·892 per cent. of fire-damp, or even if it consist only of coal-dust and pure air, it will still be so nearly inflammable that it will probably become so when it undergoes the compression and consequent heating, which the occurrence of an explosion in one part of a confined space must necessarily produce throughout the remainder of the same space. It is probable, moreover, that some kinds of coal-dust require less fire-damp than others to render their mixture with air inflammable; and it is conceivable that still other kinds may form inflammable mixtures with pure air.'

'I may mention that in the apparatus which I have hitherto employed, the proportion of coal-dust which gave the best results was much larger than might at first sight be thought necessary, namely, about one ounce of dust to a cubic foot of air for all mixtures of gas and air ranging between one of gas and forty of air. Also, in one of the experiments with the return air of a mine, the air requires to be literally *black with dust* before it will ignite. It is therefore obvious that the particles which are floating about in the air of a dry mine, in its normal state, cannot render it inflammable; and it is probable that only the sweeping action of a gust of wind like a squall, passing along the galleries, can raise a sufficient quantity to do so.'

'Some of the colliery explosions which have occurred during the last two years are amongst the most disastrous on record, and the attempts that have been made to explain them are of the usual unsatisfactory character. The assumption, without a vestige of proof, that fire-damp has suddenly burst from the strata, is still maintained even in cases in which the flame is seen to have ramified into the extremity of every *cul-de-sac* and extended to the opposite boundaries of the workings. The very token whereby the ubiquity of the flame is made manifest is the so-called *charring* of the timber, coal, and rubbish; and this, generally in the case of the timber, and always in the case of the coal and rubbish, consists of a coating of a coked coal-dust adherent to them superficially, and testifying unmistakably by its presence that coal-dust has actually been playing the part which is claimed for it by myself and others.'

'When smoke and soot are produced; or dust is ejected from the shafts; or the coal, stone, and timber have a charred appearance, due to a deposit of coked coal-dust on their surface; or, lastly, when a large superficies of the sides of the galleries is found to be on fire immediately after the event, we may safely conclude that coal-dust has played an important, if not a predominant, part in the explosion. The manner in which coal-dust operates in setting fire to coal and timber is probably as follows:—The air is travelling rapidly in one direction along a gallery, throwing a continuous shower of dust, small pieces of coal, and against all surfaces that deflect it or obstruct its course; at the instant the flame traverses it, however, the coal-dust is melted, it then assumes the properties of flaming pitch, adheres to the surfaces against which it is thrown, and rapidly accumulates until it forms a crust of greater or less thickness, according to the length of time the air continues to travel in the same direction. If it is thick enough to retain its high temperature, and is supplied with fresh air immediately, it continues to burn, and the flame soon communicates itself to the body of the coal or timber; but if it is thin, or if the surrounding atmosphere cannot support combustion, it becomes extinguished. In the second case, the surface covered with the crust or layer of coke is vulgarly said to be *charred*.'

'The dangers due to the presence of coal-dust in dry mines can be very easily avoided by sprinkling water plentifully on the principal roadways along which the air currents pass, in going to and coming from the working places. For example, Llwynypia

Colliery, which was formerly one of the driest and most dusty mines in the South Wales basin, is now kept constantly damp, or wet, in this way with a daily expenditure of about 1,800 gallons of water. The amount of air passing through it at present is over 80,000 cubic feet per minute, and its output of coal is, on the average, about 800 tons per day.'

The above extract from the paper will enable the reader to estimate the necessity of keeping the ways damp.

CHAPTER IX.

BLOWERS OF GAS.

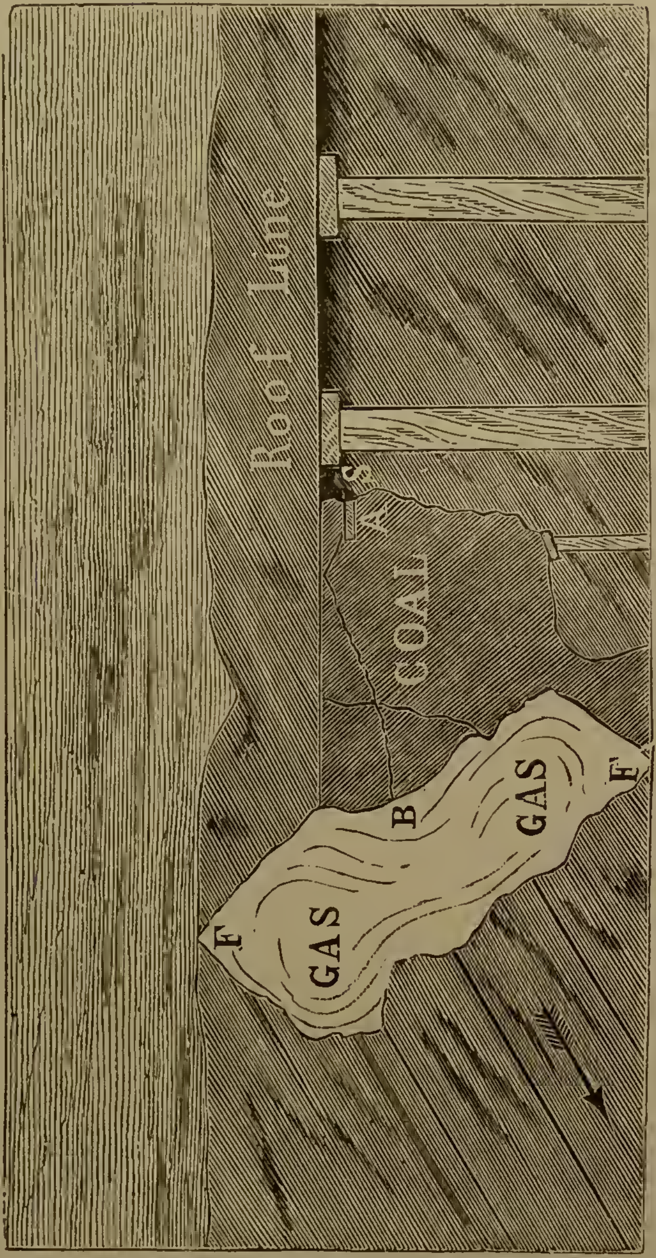
There are many seams of coal that do not emit carburetted hydrogen steadily in a regular flow, but in outbursts. It has been generally admitted that atmospheric pressure is directly concerned in such 'blowers,' as they are technically called, but some eminent mining authorities maintain that a low reading of the barometer does not necessarily induce a blower of gas. I think that when we consider that atmospheric pressure is generally represented by a pressure of 15 lbs. per square inch with ordinary barometrical readings, and since we find blowers of gas exuding from a seam with a greater pressure than 15 lbs. per square inch, we may readily assume that although atmospheric pressure is directly concerned in partially determining the amount of gas exuding slowly from the face of the coal, yet we must not imagine that because the barometer is high we are safe from a blower of gas. There are three ways in which a blower may be formed. The first is by a sudden rise of temperature expanding a body of gas confined within the seam; this cause generally brings about a feeder, or thin stream of gas exuding from the seam at a low pressure. The second cause we may assume to be due to heavy falls or subsidences of the accompanying strata, both above and below the seam in which the blower makes its appearance. The third cause may be ascribed to the displacing action of large bodies of water entering cavities filled with gas. In these latter cases the pressure will be directly as the weight of a column of water of the height, from roof to floor, of the cavity filled, unless the water is from another seam above, and will be exerted equally in all directions.

Newly opened seams are always more or less liable to small blowers; in fact, in many sinking operations, one of the first indications of the existence of the coal measures is found in bubbles of carburetted hydrogen ascending through the water in the shaft; and when such indications present themselves a centrifugal fan must be placed at the top of the shaft, and the air conveyed in canvas pipes to the bottom of the shaft to dilute the gas, and all naked lights prohibited.

A fault, especially a down-throw, is a very likely place to expect a blower; and when approaching a fault, bore-holes should be drilled 10 feet in advance of the work.

In dealing with a blower of large dimensions the following must be borne in mind:—It is *absolutely essential* that the volume, or quantity in cubic feet per minute, of the air split allotted to the particular district in which the blower exists should be so materially increased as to permit of the prompt dilution of the gas given off, and that when possible the product or combination of air and fire-damp should not be allowed to circulate in the mine, but should be conveyed in pipes directly up the upcast shaft. The gas should be encouraged by artificial means to exude; of these the simplest is obtained by driving pipes into the face or fault. The quantity of gas delivered by the blower per minute must be calculated, and the *diluting capacity* orquantity of air per minute yielded by the air split must be proportioned, as denoted in the chapter on chemical interference with the main air current, to dilute the gas. In that chapter we showed that 15 cubic feet of air will render 1 cubic foot of gas not explosive, that for further precautions the 15 cubic feet of air absolutely necessary should be increased to 100 cubic feet; then for every cubic foot of gas yielded of the blower, 100 cubic feet of air will be required over and above that afforded, under the ordinary circumstances, by the air-way. The air doors and regulators governing the splits must therefore be so arranged as to ensure that amount of air being admitted. In no case must any but self-extinguishing lamps be used in the vicinity of the blower, when, the precaution we have stated having been duly taken, the blower is reduced to a harmless evolution of gas, and may be allowed to continue as a means of draining the seam. Cavities filled with gas are the most dangerous enemy of the engineer; their existence cannot be foreseen, and their presence is too often demonstrated by a violent explosion after a shot has been fired. The accompanying plate shows how easily one of these cavities may be fired, but it must be borne in mind that pure gas will not become ignited except when its combustion is maintained by the flame being surrounded with fresh air; were it not so, on lighting the gas in an ordinary dwelling-house, the flame would travel back along the gas main till it communicated with the gas meter, which would explode, or rather become ignited.

The following woodcut will explain the necessity of putting in bore-holes when approaching a fault in driving headings with gunpowder. The shot-hole *S* has a fissure *A B* communicating with a body of gas enclosed in the cavity *F F*, caused by the downthrow shown by the arrow on the other side of the cavity. On firing the shot *S*, the gas in the cavity explodes. If the fissure *A B* contained only pure gas the flame would become extinguished for want



of pure oxygen to maintain the combustion of the gas, and no explosion would follow, but a quantity of gas would nevertheless be liberated on the coal coming down.

Large quantities of water in the sump or drainage of the mine should be avoided, as the pressure of the water due to its height in the sump hole is exerted *equally in all directions*, following a well-known law in hydrostatics: this pressure is injurious under the following circumstances. Suppose a feeder of gas is blowing in the sump; the pressure due to the water column in the sump is exerted on this feeder, which in its turn transmits the pressure elsewhere, by which means gas may blow off in other parts of the mine.

Table of Gas Pressures from Blowers.

Place	Depth in yards of seam from surface	Depth of bore-hole into ace of coal		Pressure in lbs. per sq. inch
		ft.	in.	
Harris Navigation Colliery, Aberdare 4ft. Seam	700	30	0	143
Do.	700	26	9	116
Merthyr Vale Colliery, Aberdare 4ft. Seam	450	41	0	170
Do.	450	49	9	280
Cannock and Rugeley Collieries, No. 1 Plant (Shallow Seam)	327	21	0	36
Do.	327	80	0	18

Another important feature is the pressure of the blower of gas.

The table on p. 110 gives the results obtained at three different collieries; greater pressures have been obtained elsewhere. It shows to a certain extent the necessity of avoiding 'squeeze,' or creeping, if possible, in fiery mines.

The question presents itself, when viewing these results, whether it is possible that the gas may exist within the seam in a liquefied state, through pressure. Certain gases have been liquefied under extreme pressures, and it is very difficult to account for the sudden filling of the mine with gas in so short a time. In the 9 feet 6 inches shallow seam at Cannock and Rugeley collieries the coal is very porous, which decreases the pressures that would otherwise be obtained, were the coal of a harder or more compact nature. It is here that we should recognise the danger of using non-extinguishing safety-lamps, such as Davy's or Clanny's. It is no use for competent engineers to keep pointing this out. So long as these lamps are in use, so long will there be not only explosions, but a false sense of security, which invites men to try their lamps at a blower. We consider it the duty of the Government to put a stop to the use of non-extinguishing lamps in fiery mines, and that in not doing so they have not only neglected their duty but reduced the Government inspection of mines to a very inconsistent state, bordering on a waste of time and public money. Careful engineers will not use these lamps, but at the present time there are some 12,000 in use in South Wales.

That a point in depth from the surface existed at which blowers of gas ceased, has been a theory held by many mining authorities. When divested of the superstition arising from hearsay that is so common in mining districts, the question appears to us to stand thus. If such a neutral zone does exist, then the deep seams either do not yield carburetted hydrogen, owing to the nature of their geological formation, or, on the other hand, if they do contain the excess of carburetted hydrogen that we find in fiery mines, such excess of gas is either pent up in the coal by the excess of pressure, or blows off in the shallow seams. Now, on comparing these statements with what we know to exist in English coal-fields, we find that, in the first place, it is the nature of the seam, and not its depth from the surface, that regulates the per-centage of free carburetted hydrogen. Thus the Aberdare 4-foot seam may be fairly considered a fiery seam, liable to sudden outbursts of gas. Referring to our table, we find that at a depth of 700 yards we get 143 lbs. and 116 lbs. per square inch pressure from gas. At 450 yards deep we find at another colliery in the same seam a greater pressure, viz., 170 lbs. and 280 lbs. per square inch. In the Cannock Chase district 9 ft. 6 in. shallow coal we find a pressure of 18 to 36 lbs. per square inch at a depth of only 327 yards.

This seam yields a little gas, but is not a fiery seam. Here we see that some deep seams do yield fire-damp in considerable quantities. Now we have observed that these deep seams, such as the Aberdare 4-foot, the Arley mine, the Black Vein, and many others, lying at a depth below the zone that is given as the point below which gas is not usually met with, when they do give off gas, give it off as a violent blower or outburst. It follows a heavy fall of roof or a violent creep or squeeze, and issues with a force that renders the agency of atmospheric pressure out of the question. On the other hand, the shallow seams yield fire-damp in a steady flow; the gas will be heard singing in the working faces of the coal. If the ear is placed close to the face and the seam is a wet one a slight effervescence will be heard, that is the sound produced by the bursting of the bubbles of hydrogen. These bubbles are formed by the presence of a thin layer or film of water covering the mouths of the pores of the coal, and they help us to realise in what form gas lies dormant in the coal. Violent outbursts in shallow seams are not often met with that cannot be traced to falls or squeezes brought about by winning the coal faster than the elasticity of the roof will allow it to settle. Let us imagine a colliery with three seams worked. A blower appears in the dip workings of the lowest seam with a creep or squeeze. The two upper seams are free of gas, and the roofs and floors at rest; I have seen this several times. The facts before us would lead one to look to disturbance, either chemical or mechanical, below the deep seam, to be the cause of the blower. I believe in the case of some recent explosions, both here and in Belgium, the disengagement of the gas was practically instantaneous; the mine was full of gas before any one was aware he had danger to fear. The presence of gas in a liquid form does not appear to be very probable; oxygen liquefies at a pressure of 300 atmospheres, hydrogen at 280, and nitrogen at 200 atmospheric pressures. Mechanical disturbance taking place below the deep seams does not appear reasonable enough to warrant a consideration of the probable *modus operandi.* A chemical reaction may take place by which a large quantity of carburetted hydrogen may be suddenly evolved. (We have a parallel case with pouring hydrochloric acid upon lime, only the gas evolved is carbonic acid gas.) Such chemical reaction I do not believe is possible, as if at a depth where chemical action may be supposed to be going on in the earth's crust, I think the depth would be too great for the gas to blow into our mines, and possibly the gas would become ignited.

Possibly mechanical disturbance, in the form of creep or squeeze in the strata immediately below the deep seams, is due to the same cause that produces earthquakes. These points could be more clearly explained if a bore-hole were inserted, say for 1,000 yards below the deep seam in each district, and proper gauges and instruments attached thereto to enable the observer to note the evolution of the gas; the seismograph, or apparatus to detect vibration on the earth's surface, would have to be replaced by a modified microphone and telephone to be lowered to the bottom of the bore-hole. I believe this to be the only way in which the cause of these violent outbursts of gas could be satisfactorily ascertained. Until some such experiments have been conducted, mining engineers must view the secondary feature of the case, viz., that gas cannot explode of itself, and that the artificial light used in mines shall be of such a nature that the ignition of the gas cannot be effected purposely or accidentally; further, that *pure* gas will not explode, but merely ignite and burn quietly away without detonation; that gas which is insufficiently diluted with air will detonate violently; and lastly, that gas which has an undue proportion of air will neither ignite nor explode. Good ventilation and self-extinguishing lamps will prevent this secondary dangerous property. Considerable assistance is expected to be derived in the safe working of fiery mines from the electric light. It remains to be proved whether the accidental contact of line and return wires, or the sudden fracture of a line or return wire when the circuit is complete, causing a spark of *very high* temperature to take place at the point of contact in the first instance, and at the point of fracture in the second, will not cause greater risk than the use of properly locked self-extinguishing lamps in the hands of proper men. Under these latter conditions I maintain that any coal mine is safe from explosion; and the engineer must endeavour to approach these conditions as nearly as possible, and combine them with a good and efficient means of ventilation on the basis of the laws given in the earlier chapters of this book. Under the circumstances of no Government aid to defray the cost of such experiments as I suggest, and the colliery owners not doing it themselves, mining engineers are warranted in hazarding opinions which are unsupported; as such they can do no harm, if those that advance them state that they are surmises.

The next point we have to examine in connection with blowers of gas is the atmospheric disturbance that is said to herald their approach. Most conflicting is the evidence on this point. We have a Government inspector of mines, who has kept a record of explosions and barometer readings for many years, giving it as his opinion that the results of these records do not justify him in stating that there is any relation between the two. We have the opinion of many eminent mining engineers that there is a distinct point in the reading afforded by the barometer, at which point safety ends and danger commences. The majority fix this point at 29·70.

In the face of this we have had the most disastrous colliery explosions with a high barometer reading. In the first place, from our own observation, we should say that in a great number of instances, but not the majority, considerable atmospheric, and occasionally deep disturbance does take place, at or just prior to the time of a violent outburst of gas; we have noticed that heavy rainfall preceded by excessive drought has been followed by outbursts of gas. We altogether deny the possibility of a pit firing through the discharge of lightning down the guides of the downcast shaft, and that the ascending air in the upcast shaft should be so highly charged with gas as to ignite from an electrical discharge we do not consider either probable or possible in a properly ventilated colliery. We are aware that in latitudes where earthquake is prevalent the same meteorological disturbances are observed as forming part of the phenomena which are accompanied by the earthquake. A sudden fall of the barometer does not always forewarn the approach of earthquake, but when accompanied by heavy fog, absence of wind, or cyclonic currents of short duration taking the place of the normal direction of the wind, and violent and sudden reversals of the electrical polarity of the earth, we frequently find earthquake follows. Now many old and experienced miners have testified to the 'thickness' of the air in a colliery just prior to explosion. The writer has taken the electrical polarity of the earth in a mine, and frequently found violent reversals before the barometer has fallen. It should be noticed that the connection between explosion and depression of the barometer is not worth taking into account unless it extends over at least 100 years, as powder, naked lights, breaches of special rules, &c., are very powerful factors in the question, and do not occur either in connection with the reading of the barometer or at regular intervals.

A high reading of the barometer equally shows atmospheric disturbance if the rise is sudden and irregular, and accompanied by a change of wind, and 'wind galls' making their appearance in the wind's 'eye.'

The line of a fault is the usual place for a blower to make its appearance. Some of the most experienced men who have been in latitudes where terrestrial disturbance, such as earthquakes, tidal waves, and volcanic eruption, takes place, state that the line of vibration in such cases is generally along a geological fault, and at such a point the effect is usually most severe. The concussion of a colliery explosion underground is greater than persons unacquainted with mining operations can at first realise, but no greater than the calculated effect of the expansion of the detonating mixture of gases should be; but in my experience I have never seen in examining a colliery after explosion anything sufficiently disturbed underground to admit of the theory of earthquake being the immediate cause; yet the violent disengagement of the fire-damp that does occur at times, without explosion, without either falls, creep, or squeeze, and with a steady barometer, leads me to believe that we must consider the possibility of terrestrial disturbance, whether volcanic, electrical, or chemical, as a cause of colliery explosions, before we lay the blame of every explosion on the shoulders of our mining managers and colliers in this country. This we do know, that violent fluctuations in atmospheric pressure show disturbance; that terrestrial disturbance is accompanied by such fluctuations in conjunction with violent reversals of electrical polarity, and that it takes the line of faults in the earth's crust; we also find gas more apt to blow from faults in a colliery, consequently the duty of the manager and fire triers will be to watch these faults during such disturbances as we have above mentioned. Whilst considering the subject of the sudden disengagement of gas in coal mines, it might not be out of place to remark that in the opinion of some American mining engineers, with whom the writer has been in communication on the subject, the gas is pent up by the following causes. When the coal-field was deposited geologically the expansion of the gas in the coal, with a certain outburst of spontaneous decomposition due to the heat, caused the gas to exude in large quantities from the coal, the subsequent cooling and subsidence of the bed of coal and its overlying strata causing the gas to be pent up in a compressed state. Any interference with the continuity of such a seam would cause the gas to discharge at the point of fracture, as in the case of a fault or a violent fall at the working face of the seam.

In our opinion it is idle to speculate on the causes of blowers of gas whilst powder and non-extinguishing lamps are used in coal mines together; no matter what the cause, if we have these two sources of explosions in use underground, we are not prepared to meet blowers. We urge the total abolition of these two causes of explosion, and we strongly advise managers to study to avoid unnecessary squeeze or creep whenever possible in laying out their workings. We are convinced that neglect of this is the cause of many of the complaints of non-fiery mines becoming suddenly filled with gas. The following formula will enable the reader to calculate how much gas will be given off from a goaf for a known decrease in atmospheric pressure.

Let Q = quantity of gas in the goaf, which is pent up. B = the reading of the barometer before its falls. D = the total fall of the mercury in the barometer. G = the amount of gas that will be given off.

Then—

$$G = \frac{Q\,D}{B}$$

Thus, how much gas G will come off a goaf 20 ft. × 10 ft. during a fall of the barometer from

Cannock and Rugeley Collieries.

EXPERIMENT I.

Date		Time		Pressure of gas per square inch	Barometer	Thermometer
				lbs.		°
February	19, 1881	9	A.M.	27½	30·60	46
,,	,,	1	P.M.	28½	30·60	46
,,	,,	3	,,	29½	30·65	46
,,	21	9	,,	28½	30·70	42
,,	,,	11	,,	28½	30·75	42
,,	,,	3	,,	28	30·70	42
,,	22	8.30	,,	28	30·75	61
,,	,,	12	,,	27½	30·75	62
,,	,,	3.30	,,	27½	30·70	61
,,	23	8.30	,,	27½	30·70	61
,,	,,	12	,,	28	30·70	61
,,	,,	3.30	,,	28	30·70	62
,,	24	8.30	,,	28½	30·80	61
,,	,,	12	,,	29	30·80	61
,,	,,	3.30	,,	29	30·80	61
,,	25	8.30	,,	28	30·70	62
,,	,,	12	,,	28½	30·65	61
,,	,,	3.30	,,	27½	30·55	62
,,	26	8.30	,,	28¼	30·30	61
,,	,,	12	,,	27½	30·25	61
,,	,,	3.30	,,	28	30·20	62
,,	28	8.30	,,	27½	30·30	61
,,	,,	12	,,	28¼	30·30	61
,,	,,	3.30	,,	27½	30·30	61
March	1 ,,	8.30	,,	27¼	30·50	62
,,	,,	12	,,	26½	30·50	63
,,	,,	4.20	,,	28¾	30·50	62
,,	2	8.30	,,	27½	30·55	61
,,	,,	12	,,	27	30·60	61
,,	,,	3.30	,,	26¾	30·70	61
,,	3	8.30	,,	27½	30·55	61
,,	,,	12	,,	26¾	30·50	60
,,	,,	3.30	,,	27	30·35	60
,,	4	8.30	,,	26	30	60
,,	,,	12	,,	26½	29·95	60
,,	,,	3.30	,,	26½	29·90	60
,,	5	8.30	,,	26	29·80	60
,,	,,	12	,,	26	29·80	60
,,	,,	3.30	,,	26¼	29·80	60
,,	7	8.30	,,	26¼	29·60	60

EXPERIMENT I. (*continued*).

Date	Time	Pressure of gas per square inch	Barometer	Thermometer
		lbs.		°
March 7, 1881	12 P.M.	25¾	29·55	60
,, ,,	3.30 ,,	26	29·50	60
,, 8 ,,	8.30 ,,	25¾	30	60
,, ,,	12 ,,	25½	30·05	60
,, ,,	3.30 ,,	26	30·10	60
,, 9 ,,	8.30 ,,	25	30·20	60
,, ,,	12 ,,	25¼	30·30	60
,, ,,	3.30 ,,	25	30·30	60
,, 10 ,,	8 30 ,,	25	30·40	61
,, ,,	3.30 ,,	25	30·50	61
,, 11 ,,	8.30 ,,	25	30·50	61
,, ,,	12 ,,	25½	30·50	61
,, ,,	3.30 ,,	24½	30·50	61
,, 12 ,,	8.30 ,,	25¼	30·50	61
,, ,,	12 ,,	24¾	30·50	61
,, ,,	3.30 ,,	24¾	30·50	61
,, 14 ,,	8.30 ,,	24½	30·25	61
,, ,,	12 ,,	24	30·30	61
,, 15 ,,	8.30 ,,	24¼	30·60	61
,, ,,	12 ,,	24½	30·65	62
,, ,,	3.30 ,,	24	30·70	62
,, 16 ,,	8.30 ,,	24	30·80	62
,, ,,	12 ,,	24¼	30·80	62
,, ,,	3.30 ,,	23½	30·75	63
,, 17 ,,	8.30 ,,	22½	30·90	60
,, ,,	12 ,,	23	30·90	62
,, ,,	4 ,,	22½	30·90	62
,, 18 ,,	8.30 ,,	18¾	30·85	62
,, ,,	12 ,,	19¾	30·85	62
,, ,,	3.30 ,,	20¾	30·85	62
,, 19 ,,	8.30 ,,	19½	30·60	63
,, ,,	12 ,,	18½	30·60	63
,, ,,	3.30 ,,	18½	30·55	63
,, 21 ,,	8.30 ,,	16½	30·15	62
,, ,,	12 ,,	16¼	30·15	62
,, ,,	3.30 ,,	16¼	30·20	62
,, 22 ,,	8.30 ,,	15	30·30	62
,, ,,	12 ,,	15	30·30	62
,, ,,	3.30 ,,	16¼	30·30	62
,, 23 ,,	8.30 ,,	14¾	30·20	62
,, ,,	12 ,,	15¾	30·15	62
,, ,,	3.30 ,,	16	29·95	62

EXPERIMENT I. (*continued*).

Date	Time	Pressure of gas per square inch	Barometer	Thermometer
		lbs.		°
March 24, 1881	8.30 P.M.	16¼	29·80	62
,, ,,	12 ,,	16	29·75	62
,, ,,	3.30 ,,	16	29·75	62
,, 25	8 ,,	15	29·75	61
,, ,,	12 ,,	14¾	29·80	61
,, ,,	3.30 ,,	14½	29·85	61
,, 26	8.30 ,,	15	30·15	61
,, ,,	12 ,,	15	30·15	61
,, ,,	3.30 ,,	14¼	30·20	61
,, 28	8.30 ,,	13½	30·35	61
,, ,,	12 ,,	13½	30·30	61
,, ,,	3.30 ,,	13½	30·30	61
,, 29	8.30 ,,	13¼	30·15	61
,, ,,	12 ,,	13½	30·15	61
,, ,,	3.30 ,,	13½	30·25	61
,, 30	8.30 ,,	13	30·50	61
,, ,,	12 ,,	13¼	30·50	61
,, ,,	3.30 ,,	13¼	30·45	61
,, 31	8.30 ,,	13	30·45	61
,, ,,	12 ,,	13	30·45	61
,, ,,	3.30 ,,	13¼	30·35	61

30·00 inches to 29·00 inches or 1 inch total depression?

Here Q will be the cubic contents of the cavity, or 40 × 20 × 10 = 8,000 cubic feet.

$$D = 1 \text{ inch of mercury.}$$

$$B = 30{\cdot}00 \quad \text{,,}$$

$$G = \frac{8{,}000 \times 1}{30{\cdot}00}$$

$$G = 266\tfrac{2}{3}$$

By the kindness of Mr. R. Williamson I am enabled to insert a series of observations on a blower of gas in the Cannock Chase district.

Experiment II.

Pressure at Bore-hole in West Head.

Date	Time	Pressure per square inch	Barometer	Thermometer
		lbs.		°
April 18, 1881	10 A.M.	5	30·50	51
,, ,,	11 ,,	10		
,, ,,	12 ,,	11	30·50	53
,, ,,	1 P.M.	12		
,, ,,	2 ,,	11	30·50	54
,, ,,	9 ,,	13	30·50	50
,, 19	1 A.M.	12½	30·50	50
,, ,,	5.20 ,,	14	30·50	48
,, ,,	8 ,,	17	30·50	47
,, ,,	12 ,,	14	30·50	47
,, ,,	3 P.M.	3	30·50	47
,, ,,	3.30 ,,	8	30·50	47
,, ,,	9.15 ,,	16	30·50	46
,, 20	3 A.M.	17	30·45	46
,, ,,	5.20 ,,	18	30·40	45

Note.—The first six pressures recorded were when the heading was a main intake; the remainder were when it was converted into a head-way driven with air pipes as a means of ventilation, hence the rise in temperature.

CHAPTER X.

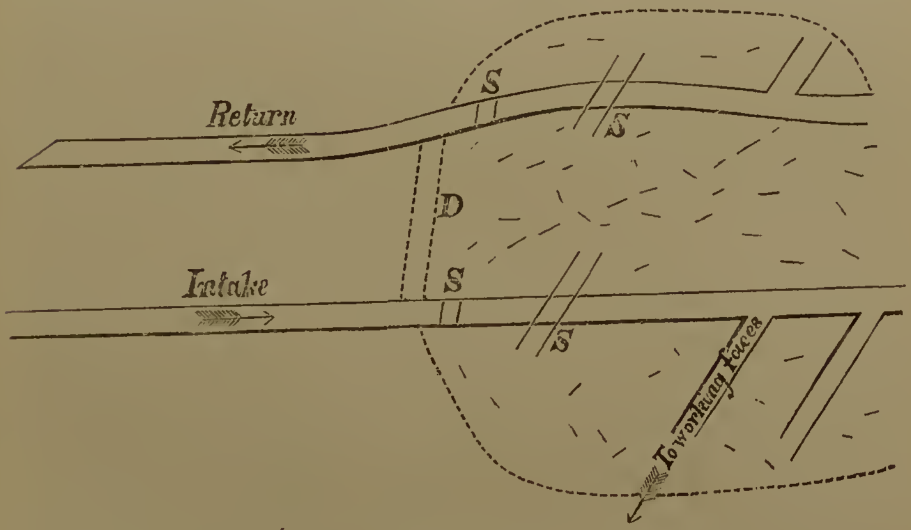
ACCIDENTS.

In a previous work on Mining Accidents I drew attention to and described several recent appliances for the purpose above mentioned. It is a lamentable fact that very few collieries, we question if any, have proper means for rescuing men from choke-damp, or carbonic oxide gas, which, as we have seen, is, if anything, more deadly than the former. In Chapter IX. of the book referred to ('Accidents in Mines,' Kegan Paul, Trench & Co., London), the result of an examination of different appliances gives Mr. Schwann and Mr. Schultz's apparatus the precedence for use in mines. Both are actuated by the principle of absorbing carbonic acid evolved from the lungs and re-oxygenation afterwards, so that, practically, the air may be breathed by the user over and over again. In Chapter XVIII. of the same work the apparatus of M. Denayrouze is fully described with instructions. Since these appliances were first brought before mining authorities, M. Fleuss's apparatus has been invented, and I consider it so efficacious that I recommend it to the notice of colliery owners. The oxygen required for the renewal of the air breathed is supplied in iron cylinders containing 30 cubic feet at 600 lbs. pressure, costing 6*d.* per foot.

These cylinders may be kept ready for use for an indefinite time. The whole apparatus is entirely self-contained, thus doing away with the serious objections and dangers arising from the use of air-pumps and conducting pipes and helmets. It will enable a man to breathe comfortably and live safely in an atmosphere of pure carbonic acid gas for more than three hours, and thus effect what he would be compelled to abandon instantly if without it. The makers of this apparatus are Messrs. Fleuss & Co., 110 Cannon Street, London, E.C.

I should advise one at least of these dresses to be kept ready at hand at collieries for use in case of explosion; and I venture to suggest a few practical hints to engineers as to the formation and operation of an exploring party after explosion, to be studied beforehand before imminent danger and necessity arise. After explosion in a colliery, and before the dust and smoke have subsided, there is a period of complete stagnation of the main air current, varying from five to thirty minutes, according to the extent of the workings and nature of the air splits affected. Immediately, at the first point of explosion, a field of choke-damp or after-damp is formed, as the product of the combustion that has taken place. It is at this moment that means may be taken with comparative safety to rescue those who may be left alive below, for these reasons :—What fire may exist in the colliery is then only commencing. The site where the heat has been greatest is surrounded by an atmosphere incapable of supporting combustion. What fire-damp there may be in the colliery has probably exploded, and the stagnation of the air current enables the return air-ways to be quickly explored before the renewal of the ventilating current has had time to sweep this foul air into them. It is, therefore, at this point that re-entry into the mine by the exploring party must be attempted. *Speed is everything.*

Dip workings must be entered from the rise, as the choke-damp will gradually work down to the dip and away from the party; but if it is on the rise in any quantity, then immediately search the dip to remove any men before the choke-damp can come down on them. Remember throughout that the *cause* of the explosion will have to be investigated, therefore bring what lamps you may find lying about with you to the surface, and take mental note of the position of any who may unfortunately have lost their lives. On feeling a renewal of the air current, I would advise the withdrawal at all events into the main intake, as near the bottom of the downcast shaft as possible, of the exploring party. At this point recollect that if there has been any fire left burning in the returns the party should leave the pit for at least twenty minutes. If no explosion follows, danger need not be apprehended from a fresh outburst of gas; but if heavy falls have been encountered, then it is not safe to re-enter the mine. Bear in mind what cannot too constantly be impressed upon every collier, that a very little fire, that a few buckets of water would extinguish in the first ten minutes, endangers the whole pit and every one in it in an hour. In bringing injured men to the surface be careful about burns, and cover the exposed surfaces of the person with oil and linen if possible. It is also desirable to provide arrangements for the surgeons to treat them temporarily at the pit's mouth before removing them home. Endeavour as far as lies in your power to separate the fatally injured from those whose recovery is possible, and especially stop all comment by on-lookers as to the cause of the accident or its ultimate result, as quiet and freedom from excitement or stimulant in excess is of vital importance. The body has undergone severe shock, the collapse of the nervous system and consequent failure of the heart is what must be guarded against, and the treatment *for the first few hours* after an accident is of vital importance. To carry out these particulars it is necessary to divide your resources into three: those that explore the workings, those that take the injured up, and those that attend to them when up. One of the most important features to be recollected in finding a man conscious but relapsing into a semi-comatose state is to ascertain from him when he last had food, and to remind the surgeon accordingly. One-fifth the stimulant required for a man after dinner would suffice for restorative purposes if given before. Whoever is in command of the two underground parties should keep up constant communication between the two, so that a man returning into the workings need not re-enter alone. So far the engineer can try and mitigate the after consequences of explosion to those who are injured. Two important points will present themselves, however, in all probability, for his ultimate and occasionally immediate consideration: the extinction of fire in the workings, and the renewal of ventilation. If the fire is great, close the workings by stoppings, not the shafts. If adjoining a mine with separate shafts, entry must be effected through that mine to the one that is on fire, taking care to protect the entrance on either side with stoppings that can readily be fitted up if required. If the stoppings in the pit which is on fire are good there is no danger in setting the fan going again, by making a new drift from the intake near the stoppings into the return, thus :—



D is the new intake.

SS are stoppings.

The area as shown by the dots and dotted line is supposed to represent the portion of the mine in which the explosion has occurred and the fire exists.

We practically construct a new mine by excluding that portion which is affected and dangerous. It will be found that it is the returns that are the most difficult to penetrate after explosion; and the longer it is delayed the worse it is, therefore we recommend their being examined and searched first.

In the event of its being necessary to close the shaft use wet sand, not clay, and concrete over it.

Circumstances only can dictate the advisability or not of readmitting air after explosion, and no general rule can be laid down on this point. The great difficulty that always presents itself on entering a mine is falls; and it is most important to try and ascertain whether they have occurred before or after the explosion, as if they occurred before explosion the cause may possibly be found to be a defective lamp in the vicinity, or a Clanny lamp broken by the fall itself. As an instance of the importance of examining lamps after explosion, I was asked to examine some lamps after a fatal explosion, in which the manager was killed. Many lamps were broken, but one had a quantity of mud blown up *inside* the gauge and glass; the top was off. On examining this lamp the marks of wiping the *inside* of the top were plainly visible; the bottom of the lamp was found close to the body of the person who owned it. I have little doubt that he was engaged in wiping the lamp top clear after having dropped it in the mud, having left the bottom piece burning, and so the gas became ignited.

In the event of the apparatus I recommend being kept at a colliery ready for use the men must be drilled to use it. Of course the circumstances attending accidents at a colliery are very variable, and I simply wish in these few points to draw the attention of those engaged in collieries to what features in my experience as a mining engineer are likely to be overlooked in the confusion and consternation that must prevail immediately after an explosion; it is a moment of the greatest anxiety to the manager, and he will be better prepared to do his duty when thus suddenly called upon, if he studies the possible surrounding circumstances of an explosion before it occurs. One more feature must be recollected. In despatching messengers it is most essential to give only written instructions addressed to those for whom they are meant. Messengers from a colliery where an explosion has occurred are sure to be stopped and interrogated by persons, and thus unaccountable and serious delays occur. With the underground parties, confidence in their engineer and previously concerted action are imperative; for the medical staff, room and silence, and for the lookers on a good staff of mining police to clear them off the ground; but it is advisable always to keep them informed of what is going on, and thus prevent them from spreading false news, and from interviewing those who have been severely injured and have been conveyed to their own homes. The measure advocated here keeps them quietly talking amongst themselves and out of the pit village till night, when the majority generally disperse quietly to bed.

Precautions against accident from foul air, and means to be taken after to restore the injured.

All underground men are liable to be asphyxiated or 'choked off' when entering abandoned workings, sump-holes, furnace drifts, or more especially after explosion or during gob-fire.

Observe the following :—

1. Immediately withdraw the persons affected, and place them in a pure atmosphere out of the mine.

2. Undress them and bathe their bodies with cold water.

3. Make them swallow a little water slightly acidulated with vinegar.

4. If this fails to restore consciousness, administer a clyster of $\frac{2}{3}$ cold water and $\frac{1}{3}$ vinegar.

5. If this fails administer a clyster of weak solution of common table salt, or if procurable Epsom salts for preference.

6. Irritate throughout the lining of the nostrils very gently with a quill feather.

7. Blow air up one nostril and close the other

with the finger. A common pair of bellows will answer well.

8. If these fail in rotation to revive the patient, and the eyes appear suffused, bloodshot, and the eyelids swollen, or the lips are purple and swollen, open a vein in the foot and withdraw not more than a wineglass of blood. Remember to stop the bleeding when the amount has been obtained.

9. Failing this adopt artificial respiration. Do not suppose that apparent total inanition with inaudible action of the heart indicates death.

Artificial respiration should be practised by the subordinates of the colliery *before* occasion arises. The Royal Humane Society of London have published a chart with illustrations descriptive of the system, and the colliery should be properly provided with these *above and below ground.*

Note.—That restoratives, stretchers, burn mixture (whiting and sweet oil), cotton wool, &c., should be kept *in the pit*, as well as above ground.

Failing a stretcher, take up a short length of the tram road to convey the injured person to the pit bottom.

Avoid giving stimulants too freely when amputation may subsequently be necessary.

Entrance into choke-damp is facilitated by holding a piece of cotton waste saturated with solution of Glauber salts in front of the nose and mouth, but it is essential to oil the face where the cotton waste will touch the skin first, or the solution will burn the skin of the lips.

Measures to be taken in the event of an accident occurring in mines, quarries, &c., in France and Belgium (Government rules).

Act 11. In the event of an accident occurring, master miners and overlookers are bound to give notice of such accident to the mayor, the engineer of the mines, or in their absence to the manager.

Act 12. The same obligation holds good where the safety of the works below or buildings on the surface is endangered.

Act 13. In every case the engineer will proceed to the spot, and will draw up a separate statement of the accident, giving the cause, and will forward it to the Prefect of the Department.

Act 14. When the mayor or the police have been informed of an accident having taken place, whether such information is derived from the explorers or from the public, they will advise the superior authorities of such accident, and will take such precautions conjointly with the engineer as shall stop all immediate danger and prevent after consequences. To this end they are empowered to make requisition for horses, men, and material, and will issue the necessary orders, the execution of which will be carried out under the direction of the engineer, or in his absence the workmen will be directed by picked men from the district.

Act 15. The mine owners are compelled to keep on the colliery property, surgical and medical appliances, in proportion to the number of men employed, the nature of which appliances shall be decided by the Minister of the Interior, and they will comply with the code of rules drawn up and approved by him.

Act 16. The Minister of the Interior, on the recommendation of either the Prefect or the Director-General of Mines, will decide what mines (on account of the large number of men employed, or by reason of special circumstances) are to keep a resident surgeon, who may be attached to a district.

Act 17. Directors of mines and others near the scene of an accident will do all in their power to assist with men and material.

Act 18. The cost of such or other assistance given will be defrayed by the owners of the mine where the accident happens.

The sum of these rules is in our opinion calculated to cause delay by those who are in a responsible position endeavouring to comply with them, instead of setting to work to rescue the men and repair the damage.

Such stress is laid by some mining authorities on the observation of the weather, that we give a summary here of the laws affecting changes of pressure in the atmosphere.

CHAPTER XI.

METEOROLOGICAL PHENOMENA AFFECTING COLLIERIES.

The humidity of the air has a direct effect upon the presence of coal-dust in collieries. Professor Abel, in his experiments on samples of dust taken from Seaham Colliery in 1881, confirms Mr. Galloway's experiments, and states that even calcined magnesia will deflagrate when in suspension in air. It is advisable therefore to use an hygrometer in dusty mines. The wet bulb thermometer is the best form of the apparatus. Messrs. Apps make a special portable form for mines. The evaporation of water in the cup and over the bulb produces cold. This thermometer, therefore, will register lower than the average temperature of the mine recorded by the dry bulb thermometer; the difference between the two readings represents *the power the air possesses of evaporating water.* This power will vary in accordance with the degree of humidity, or amount of water in suspension in the air, therefore the evaporative power thus indirectly becomes a means of measuring the humidity of the air in the mine.

By multiplying the excess of the dry bulb over the wet bulb thermometer by the factor, the excess of the temperature of the air above that of the dew point will be ascertained. For convenience' sake the following table is inserted. D is the reading of the dry bulb thermometer.

D	Factor	D	Factor	D	Factor
10°	8·78	40°	2·29	70°	1·77
11	8·78	41	2·26	71	1·76
12	8·78	42	2·23	72	1·75
13	8·77	43	2·20	73	1·74
14	8·76	44	2·18	74	1·73
15	8·75	45	2·16	75	1·72
16	8·70	46	2·14	76	1·71
17	8·62	47	2·12	77	1·70
18	8·50	48	2·10	78	1·69
19	8·34	49	2·08	79	1·69
20	8·14	50	2·06	80	1·68
21	7·88	51	2·04	81	1·68
22	7·60	52	2·02	82	1·67
23	7·28	53	2·00	83	1·67
24	6·92	54	1·98	84	1·66
25	6·53	55	1·96	85	1·65
26	6·08	56	1·94	86	1·65
27	5·61	57	1·92	87	1·64
28	5·12	58	1·90	88	1·64
29	4·63	59	1·89	89	1·63
30	4·15	60	1·88	90	1·63
31	3·70	61	1·87	91	1·62
32	3·32	62	1·86	92	1·62
33	3·01	63	1·85	93	1·61
34	2·77	64	1·83	94	1·60
35	2·60	65	1·82	95	1·60
36	2·50	66	1·81	96	1·59
37	2·42	67	1·80	97	1·59
38	2·36	68	1·79	98	1·58
39	2·32	69	1·78	99	1·58

Cyclonic action of the wind is a great source of disturbance to the barometer. The cause of gyratory or cyclonic winds may be thus explained. The general contour of land offers, by its valleys and mountains, considerable opposing and deviating obstacles to the ordinary circular motion of a storm.

On the sea such obstacles do not exist, hence the prevalence of such storms at certain seasons of the year, as the motion of cyclones is twofold. One, the gyratory motion around its centre: north of the equator this gyratory motion is from right to left, whilst south of the equator the motion is reversed, or from left to right. The other motion is that of translation. The speed varies from 35 to 10 miles per hour. The path of a cyclone in the Northern Hemisphere within the region of the trade winds is towards the west, inclining towards the north. In latitude 20° the path is more northerly, till in latitude 25° it becomes nearly N.W.; at 30° latitude it is due north, after which it begins to veer in an easterly direction.

In the Southern Hemisphere the motion is exactly reversed.

The barometer is lowest near the centre of the storm area. The duration of the storm will depend upon its diameter. The centre of the majority of the severe storms follows the north boundary of the United States. The general appearances of the sky, as recorded by the United States Signal Service that predict 'weather,' may be thus summed up.

Clouds above those nearest the earth, when moving in an opposite direction to the wind, foretell a change in the direction of the wind. Remarkable clearness of the atmosphere indicates rain. Halos, coronæ, show the precipitation of vapours in the higher regions of the atmosphere, and are followed by rain or snow.

When the outlines of the large clouds called cumuli are clearly defined, fine weather may be expected. The electricity of the air is seldom negative except when rain is falling, and the colder the air the greater the intensity of the charge.

In the ventilation of a fiery mine I advise the quickening of the fan a few strokes during dull and foggy weather, or during the depression of barometrical pressure that generally precedes the equinoxes.

Explanation of the Gas Chart.

The object of this chart is, firstly, to show the method to be employed in making observations on blowers or feeders of gas in a fiery mine, when danger may be apprehended; and secondly, to illustrate the effect of changes of temperature, and pressure in the volume of the gas. The thickness of the gas was obtained by erecting a 60-inch staff, graduated in inches, and placed where the gas showed itself; the fire trier slowly raised his lamp until the 'cap' showed on its flame, when the level of the flame, read off on the staff, gave the equivalent level of the gas. The observations were taken three times a day, viz., at 9 A.M., 12 noon, and 3 P.M., the pit being running, on the 23rd. The greatest thickness of gas was 4 ft. 10 in. in the 5-ft. heading, leaving only 2 inches of air on the bottom of the roadway. The atmospheric condition in the pit was shown by the instruments to be dry, 4° Fahr. in point of temperature above the surface temperature, and higher than the normal temperature of the pit, rising towards noon, while the atmospheric pressure was on the surface 28·9 inches, equal to a pressure per square inch of 14 lbs. $7\frac{1}{5}$ oz., and in the pit 29·3 inches, equivalent to a pressure of 14 lbs. $10\frac{2}{5}$ oz. per square inch (being a deficiency of pressure per square inch of rather more than three ounces avoirdupois from the normal atmospheric reading of 15 lbs. per square inch on a barometer reading of 30 inches).

On the other hand, the minimum thickness of gas was on April 18 at 12 noon, when it was only 1 ft. 7 in. in the 5-ft. heading, the surface temperature being 42° Fahr., and that in the workings 58°; the barometer reading in the workings being 29·6 inches, or 14 lbs. $3\frac{1}{5}$ oz. per square inch, and on the surface 29·1 inches, or a pressure per square inch of 14 lbs. $2\frac{4}{5}$ oz. In the chart, No. 1 barometer was situated on the surface, No. 2 barometer in the pit bottom, No. 3 in the workings. The thermometers are thus shown: No. 1 thermometer was on the surface, No. 2 in the pit bottom, No. 3 was 618 yards in-bye, No. 4, 1,236 yards in-bye, No. 5 in the workings. The condensed results of the observations when tabulated are as follow:—

1879	Fan P.	Face	Atmospheric pressure				Hygrometer showed pit to be	Thickness of gas in heading	
			In pit		On surface				
	°	°	lbs.	oz.	lbs.	oz.		ft.	in.
April 17 .	57	41	14	12	14	$8\frac{4}{5}$	wet	3	1
,, 18 .	58	42	14	$14\frac{2}{5}$	14	$8\frac{4}{5}$	dry	1	8
,, 19 .	58	48	14	$12\frac{4}{5}$	14	$8\frac{4}{5}$	dry, then wet	2	0
,, 21 .	60	47	14	$11\frac{5}{6}$	14	9	wet, then dry	4	2
,, 22 .	58	47	14	$11\frac{1}{5}$	14	9	dry	4	2
,, 23 .	58	52	14	$10\frac{2}{5}$	14	9	wet, then dry	4	10
,, 24 .	57	47	14	$13\frac{3}{5}$	14	$8\frac{4}{5}$	wet	3	2
,, 25 .	57	53	14	$14\frac{2}{5}$	14	$10\frac{2}{5}$	dry, then wet	2	9
,, 26 .	57	50	14	12	14	8	dry	4	5
,, 28 .	58	47	14	$14\frac{2}{5}$	14	$11\frac{1}{5}$	dry	4	6
,, 29 .	56	54	15	$\frac{4}{5}$	14	$12\frac{4}{5}$	dry	3	1
,, 30 .	57	50	15	—	14	$12\frac{4}{5}$	dry	3	0
May 1 .	58	46	15	—	14	12	dry	2	11
,, 2 .	59	43	15	$9\frac{3}{5}$	14	$13\frac{3}{5}$	dry	2	4

It is impossible, from these data, to say exactly what effect the condition of the atmosphere has on the *quantity* of gas in a mine; it must be obvious to the reader that the difference of a few ounces in pressure per square inch—and that is all we can record by the barometrical changes—cannot affect a blower of gas issuing with a force of possibly twice the pressure of the atmosphere. The question resolves itself purely into one of equilibrium; on the one hand we have a constant force exerted by the nucleus of air by which the earth is enveloped, varying, when estimated in pressure per square inch, from 14 to 15 lbs.; on the other hand, we have a discharge of gas issuing with a pressure of from 1 lb. to an unknown limit per square inch; the obvious deduction is this up to 15 lbs.; that is, that so long as the barometer stands at 30 in the pit, the atmospheric pressure counteracts 15 lbs., and only that amount, of the pressure with which the blower issues; that if this latter pressure is more than 15 lbs., the origin of its excessive force must be attributed to some other source than any abnormal decrease in the surface barometrical readings.

The observations illustrated by the chart are the ones I have selected from a considerable series of experiments, as being a fair specimen of the average results yielded.

With violent outbursts of gas in mines, whether normally fiery or not, experience leads us to the following conclusion. There is less risk of outbursts with long wall work, as there are no portions of coal left, the coal is cleared out now and the roof comes down gradually as the work proceeds, and packs up better, leaving no goafs, and relieving the 'crush.'

One thing, the importance of which in a district liable to blowers cannot be overlooked, is that the roof must come down well; any shale or bind must be ripped off and the solid roof got to before allowing it to come down and pack up, otherwise when the bind comes down (before the roof), the cavity between the bind and the real rock roof will be filled with gas. When the roof comes down the gas will come with it right on to the floor of the workings and fill them. In support of this let us examine the cause of the awful catastrophe at Abercarne in 1878.

Here the air current was 23,560 cubic feet per second. A fall took place in No. 2 district that filled that district temporarily with gas until swept away by the current of air circulating in that district. It was found on examination that a considerable portion of the roof had fallen up to a parting four feet above the coal, and between this parting and the bed of rock above there was a cavity 1 foot deep, extending over a considerable area, forming the reservoir from which the gas had come; the lamps of the men were extinguished, and no calamity occurred that day. However, almost simultaneously with this outburst at Abercarne, it was discovered that on the east side long-wall workings on the old coal in No. 15 pit, all the fissures and cracks in the roof down to the coal were full of gas, and similar cavities were found as at Abercarne. A gas drain was immediately put in on the rise side of the coal, when the gas disappeared.

When falls occurred afterwards in the pit it was found that these cavities did not hold gas, it having escaped by the drain.

There is no doubt that it is best to work long-wall way in any seams that are not thicker than can be taken at once. Pillar and stall work is more suited to such seams as the 10-yard coal in South Staffordshire. When the seams dip, as in North Staffordshire, it is best to work from the rise to the dip; this lessens the danger of blowers.

The Ebbw Vale Company use a gas extractor for trying the cavities when the firemen cannot insert a lamp, and it is reported to answer extremely well.

There appears to be a greater probability of gas blowing at the commencement of frosty weather.

To estimate the amount of gas that will be yielded for a difference of atmospheric pressure from a goaf or cavity, Marriott's law that the volume varies inversely as the pressure is the simplest in practice.

For instance, the barometer reads 30·00 inches; it falls to 29·50 inches. It is required to find the increase in volume of the gas given off from a goaf or blower. Then 29·50 : 30·00 : : volume at 30·00 inches : volume at 29·50 inches.

Or (volume at 29·50) — (volume at 30·00)

= volume of gas that will leave the coal or goaf.

The elasticity of hydrogen increases more rapidly than the pressure. The possibility of liquefaction diminishes as the elasticity of the gas, as compared with other gases, increases.

Elasticity of Gases at High Pressures.

Pressure in lbs. per square inch	Air	Nitrogen	Carbonic acid	Hydrogen
15	1·000	1·000	1·000	1·000
150	9·916220	9·943590	9·226200	10·056070
350	19·719880	19·788580	16·705400	20·268270

Gas	Density	Square root of density	$\frac{1}{\sqrt{\text{Density}}}$	Velocity of diffusion air = 1	Rate of effusion
Hydrogen . .	0·06926	0·2632	3·7994	3·83	3·613
Marsh Gas . .	0·559	0·7476	1·3375	1·344	1·322
Carbonic Oxide .	0·9678	0·9837	1·0165	1·0149	1·0123
Olefiant Gas .	0·978	0·9889	1·0112	1·0191	—
Sulphuretted Hydrogen . .	1·1912	1·0914	0·9162	0·95	—
Carbonic Acid .	1·52901	1·2365	0·8087	0·812	0·821

The rate of effusion is the rapidity with which the gases would escape into vacuum through the minutest hole, say $\frac{1}{300}$ of an inch.

CHAPTER XII.

SAFETY LAMPS.

On this subject so much has been written, and so many merits claimed by each inventor, that without prejudice to any I propose to divide the subject into the following headings.

Firstly, what lamps are used in coal mines, then the objections raised against these by those who have used them, and finally, the sum of my own experience with them.

The safety lamps most generally used in England are Mueseler, Clanny, Davy, Stephenson, Williamson's Patent Double Safety Lamp, and the Protector Lamp.

Of these Mueseler, Stephenson, and Williamson's lamps are 'extinguishing;' that is, they possess the property of automatically becoming extinguished in an explosive atmosphere, a point of vital importance when blowers exist. Clanny's, Davy's, and the Protector lamp do not become extinguished. I refer the reader to a paper on this subject read by me on April 25, 1879, before the Institution of Mechanical Engineers in London, and published in their *Transactions.*

Diagrams are given in this paper of the principles of the combustion chamber in each lamp. I consider non-extinguishing lamps in fiery mines unsafe, and contend that there is ample evidence to support this rule. A blower of gas of 8 feet per second velocity will pass a Davy lamp and become ignited. The question to be decided is, Can a miner extinguish his lamp sufficiently quick when he discovers the gas burning inside it to prevent explosion? I contend that he cannot. This disposes of all single-gauze safety lamps on Davy's principle. Let us quote the report of the Mining Accident Commissioners on this point, who examined a great number of witnesses both for and against the use of these lamps. The experiments were made at Garswood Hall Colliery by means of a powerful blower of gas which existed there. Accurate velocities and percentages were obtained by delicate and accurate scientific apparatus. Pending further experiments the commissioners report, '*One result was so distinctly elicited, confirmatory of what had been announced by others, that it was thought right, pending the prosecution of the inquiry, to announce it to the Secretary of State for the Home Department. It is to the effect that the employment of the ordinary Davy and Clanny Lamps in an explosive mixture, when the current exceeds six feet per second, is attended by risk of accident almost amounting to a certainty.*'

Nothing can be clearer than this point. It is one which I have insisted on for more than three years and to no effect, and until these two lamps are condemned and their use made illegal, managers will still use them and incur the risk. I can only suggest that any engineer who uses Davy or Clanny lamps uses them now well knowing the risk. In the hands of the fireman I do not object to them, provided they are surrounded by an outside glass cylinder. Mr. R. Forster, manager of the South Hetton Collieries, reports an explosion with the Protector mineral oil lamp, but not from gas, from the paraffin oil itself. Mineral oil is highly objectionable in a colliery. Mueseler's lamp usually becomes extinguished, but in the experiments at the end of this chapter it will be noticed that it occasionally explodes. This feature has been cleared up thus. If the width of the orifice at the top of the inner chimney is too great there is a risk of fresh air blowing down it on to the flame, and so keeping the wick alight till the gas passes the gauze or destroys it. It stands a very high velocity, but in the hands of men engaged in timbering and setting collars it is useless, as it frequently goes out if tilted to one side. An examination of the evidence of men who have used the lamp will support this.

Stephenson's lamp gives too little light, otherwise it is safe. I find no valid objections against Williamson's lamp, and it is absolutely safe.

The sum of my experience with safety lamps is, that many of those who use them, *i.e.* the miners, do not understand how to take care of them, nor how to use them in gas. Every lamp should be tested in gas before it is sent into the pit, and there should be no lamp trimming stations underground except in the intake air-current; indeed, even there it is most desirable to have the station as near the downcast shaft as possible. I think there are the gravest objections to the system of colliers buying their lamps. It generally ends in a collection of safety lamps of all patterns and ages, and frequently causes disputes.

I have continually pointed out the danger of using non-extinguishing lamps in fiery mines, and at last the attention of the Government has been called to the danger by the commissioners, but great blame attaches, to my mind, that this fact was ignored so long. The qualifications for a good safety lamp are as follows :—

a. In an atmosphere of fire-damp at too high a velocity the lamp should become extinguished and should not explode.

b. The lamp should be able to stand a high velocity without becoming extinguished too soon. It should also be a delicate lamp in the hands of the fire-trier.

c. The body of gas burning within the lamp should not continue to burn after the lamp goes out.

d. The lamp should be able to be inclined at a considerable angle, and yet not go out.

e. It should not be too heavy, should burn nothing but vegetable oil, and should yield a good light in any position.

All these requirements Williamson's lamp fulfils.

Toughened glass is unsafe for safety lamp glasses. It frequently breaks of its own accord.

So far electricity has not been practically applied to lighting coal mines. No doubt it will ultimately be employed successfully, but it must always be borne in mind that hermetically sealed electric lamps, although they yield an excellent light, do not indicate the presence of fire-damp, and that consequently the watch that is kept by the use of safety lamps throughout the workings of a large coal mine on the sudden blowers of gas, will be removed if safety lamps are not also employed in addition to the electric lamps.

Subjoined we give details of experiments with safety lamps. The results all tend in the same direction, namely, the condemnation of Davy's and Clanny's lamps, and all non-extinguishing lamps. We have never either recommended or permitted their use when responsibility rested with the author, and we recommend mining engineers who are doubtful on this point to experiment for themselves, and not to have these lamps if they find them 'explode' in explosive mixtures at high velocity. Subjoined are some tables of experiments with safety lamps.

A few practical hints may be added as to how to estimate the per-centage of air in combination with fire-damp with the lamp, when looking for gas. It is a known fact that if fire-damp contains an excess of bicarburetted hydrogen, a red-hot wire will fire it; again, the force of an explosion of fire-damp will be directly proportional to the amount of oxygen it contains to ensure combustion. One cubic foot of gas to five and a half cubic feet of air is the most feeble combination; the gas if ignited merely 'flashes.' The maximum effect is reached at 1 of gas to 10 volumes of the air; this point is shown by a blue 'cap' or halo round the top of the flame. If a Davy or Clanny lamp is ever immersed, the flame instantly begins to mount the inner gauze after the 'cap' has made its appearance. To observe the 'cap' the direct rays from the light must be shielded from the eyes, and this is best done by observing the top of the flame between the two fingers. Fall of roof, faults, and all abandoned workings must be ap-

Cannock and Rugeley Collieries, March 1, 1878.

EXPERIMENTS MADE WITH SAFETY LAMPS, AS UNDER:—

No. of Experiment		Description of Lamp	Velocity of air per second	Length of time for each Experiment	Result of each Experiment
No. 1 Experiment	1	Davy	8 Feet	20 Seconds	Exploded
	2	Clanny	do.	120 ,,	No Explosion
	3	Mueseler	do.	56 ,,	Extinguished
	4	Stephenson	do.	8 ,,	do.
	5	Williamson	do.	9 ,,	do.
No. 2 Experiment.	6	Davy	10 Feet	16 Seconds	Exploded
	7	Clanny	do.	65 ,,	do.
	8	Mueseler	do.	14 ,,	do.
	9	Stephenson	do.	1 ,,	Extinguished
	10	Williamson	do.	1 ,,	do.
No. 3 Experiment.	11	Davy	16 Feet	8 Seconds	Exploded
	12	Clanny	do.	24 ,,	do.
	13	Mueseler	do.	4 ,,	Extinguished
	14	Stephenson	do.	1 ,,	do.
	15	Williamson	do.	1 ,,	do.
No. 4 Experiment.	16	Davy	20 Feet	10 Seconds	Explode
	17	Clanny	do.	9 ,,	do.
	18	Mueseler	do.	11 ,,	do.
	19	Stephenson	do.	1 ,,	Extinguished
	20	Williamson	do.	3 ,,	do.
No. 5 Experiment.	21	Davy	25 Feet	5 Seconds	Exploded
	22	Clanny	do.	61 ,,	do.
	23	Mueseler	do.	5 ,,	Extinguished
	24	Stephenson	do.	4 ,,	do.
	25	Williamson	do.	4 ,,	do.
No. 6 Experiment.	26	Davy	30 Feet	6 Seconds	Exploded
	27	Clanny	do.	10 ,,	do.
	28	Mueseler	do.	6 ,,	do.
	29	Stephenson	do.	2 ,,	Extinguished
	30	Williamson	do.	1 ,,	do.
No. 7 Experiment	31	Davy	16 Feet	13 Seconds	Exploded
	32	Clanny	do.	7 ,,	do.
	33	Mueseler	do.	2 ,,	Extinguished
	34	Stephenson	do.	25 ,,	do.
	35	Williamson	do.	15 ,,	do.

proached with care; the existence of a blower should never be searched for with any but an extinguishing lamp on Stephenson's principle. The sump should be carefully tried for gas occasionally: if it contains any gas, a slight rise in the water level will make the gas blow through the covering; and since the cager generally works with a naked light an avoidable source of danger is incurred, and a watch must therefore be kept on it. When working a very fiery seam, the rate of progress must be sufficiently slow to allow the gas to drain off quietly. In conclusion, the *only* system by which a fiery mine can be worked safely is by paying strict attention to what appear at first sight to be minor details. The fire-triers must be picked men, and those men who are connected with the ventilation department must be practically acquainted with their duties. We recommend every viewer to examine the general effect of the main air-current periodically. He should ascertain how much air, at what velocity, is required, according to the rules we have here given, and his duty is to see that (i.) the amount of air passing into the working is *adequate*; (ii.) that the adequate quantity does really pass through the pit.

No instructions to subordinates will avail, unless the chief engineer satisfies himself that the instructions are necessary; and if so, that they are carried out in the spirit with which they were issued. The ventilation of a colliery is a most responsible duty, and since the laws that govern the velocity and quantity of the air-current are now understood, the responsibility is reduced to a minimum, but if neglected, explosion is the certain result. More explosions have occurred through neglect to ascertain what the effect of splitting or stopping would be *before doing it* than through insufficient ventilation. A prevalent idea amongst roadsmen is that if a large quantity of air passes down the shaft, the pit is well ventilated. This entirely depends on the disposition of the air-current in the pit; and further, if the shafts are calculated to yield a passage to a given volume of air, and the requirements of the mine are such that that volume is absolutely necessary, the shaft must not be blocked up with guides, signal wires, pump rods, &c., after the calculation is made; and in designing new shafts possible contingencies such as electric light and compressed air pipes, &c., should be allowed for when calculating the shaft room.

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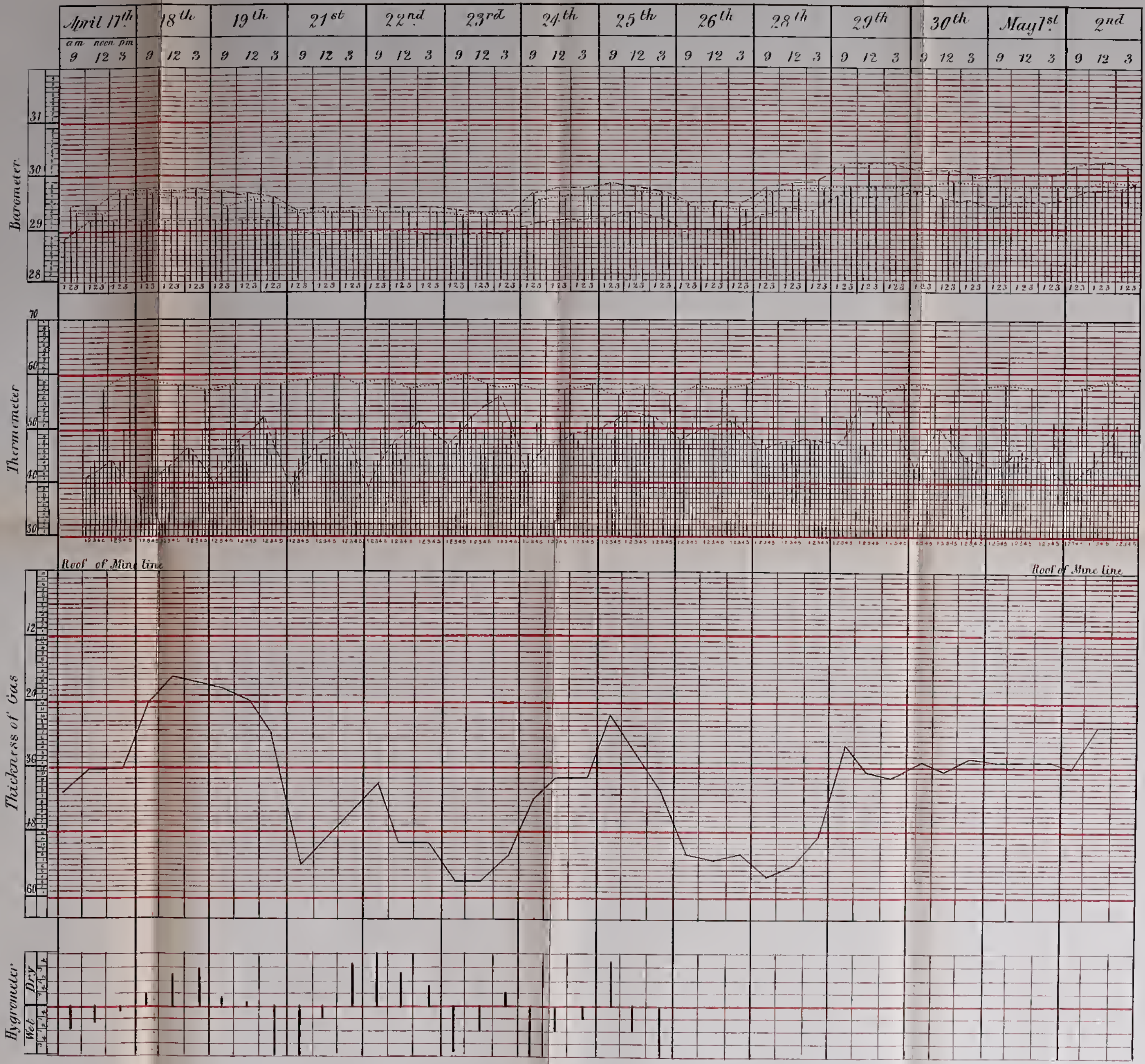
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